Metallurgical and Chemical Engineering

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CONTENTS.

Trends in Iron and Steel Production.

As most of the important statistics of iron and steel production in the United States in 1910 have now been reported a glance at the results of the year may be of interest. The pig iron output was 27,298,545 tons, and while it broke the previous record, made in 1909, by 3 per cent it was much below the capacity. Indeed, in the twelvemonth from July 1, 1909, to July 1, 1910, production was practically 30,000,000 tons, and conditions were such as to leave room for the belief that in the calendar year between 32,000,000 and 33,000,000 tons of pig iron could have been made. At the lower estimate the actual production was 15 per cent short of the capacity, even though a new record in output was made.

The lead of Bessemer steel over open-hearth, first established in the "off" year of 1908, was more firmly established, as in 1910 production of open-hearth steel ingots exceeded that of Bessemer by precisely 75 per cent, while the lead of basic open-hearth over Bessemer, disregarding acid open-hearth entirely, was 62 per cent.

While the production of electric steel was only the modest quantity of 52,141 tons, this exceeded the production of any other country. As the introduction of this method is very recent, rapid gains in tonnage are to be expected for this year and next.

Without any apparently heavy demand for steel rails the production was 3,634,029 tons, thus exceeding the production of all previous years with the exception of 1906. The production of rails is not destined to see the violent fluctuations which have occurred in the past, when the bulk of the rails purchased were either for the building of new track or the replacement of old and light rails which the rapid adoption of heavy cars and locomotives suddenly made unfit. A large part of the annual rail demand is now due to the wearing out of rails in ordinary service, and as railroad traffic does not vary greatly from year to year the production is correspondingly uniform.

The production of open-hearth rails closely approached that of Bessemer in 1910, the total of 3,634,029 tons being made up of 1,917,900 tons of Bessemer, 1,715,899 tons of open-hearth and 230 tons of iron rails, so that Bessemer production was 53 per cent of the total. The production was much less than the capacity, which may be stated at 7,000,000 tons if all possible equipment be included. It should be observed, however, that many of the rail mills are interchangeable, rolling billets or sheet bars in place of rails upon occasion, and when they are engaged on these products merely the rail finishing department is idle. This flexibility was long ago found to be very desirable on account of the wide fluctuations in rail demand from year to year. It would not be difficult to make nearly as large a tonnage of open-hearth rails in a year as was made of all rails last year, but it is not at all certain that this will be

done. There is still a definite demand for Bessemer rails, and, of course, the difference of $2 a ton in the market price is not an unimportant factor with many roads, particularly for track over which the traffic is relatively infrequent.

The Power Problem for Electrochemical Plants.

In any new electrochemical enterprise, as in any new industrial undertaking, the concrete question always comes up at an early date, "where shall we locate?" It is almost tautology to say, "at the place where we can make the most money." With respect to electrochemical processes an impression prevails that the largest single factor in determining the cost of operation is the cost of power. This is undoubtedly true as a general rule, but at the same time it is true that there are a great many other factors which play an important rôle, and all must be balanced and properly proportioned. When this is considered, the availability of large amounts of cheap hydroelectric power for the electrochemical plant will not be found so absolutely necessary as a cursory examination of the subject might indicate. This is especially true in the case of conditions in the United States. In brief, the location of the electrochemical plant is a question of commercial geography. Besides the problem of power, there must be considered the conditions of the labor market, of shipping facilities by rail or water, of freight rates, of supply of raw materials and fuel and of the market for the finished product.

Each of these elements has its bearing on the final result and each has its own factor of commercial efficiency. In any multiplication of efficiencies each factor, large or small, counts toward the final result; 60 per cent multiplied by 70 per cent by 80 per cent gives a higher efficiency than 95 per cent by 90 per cent by 30 per cent. The balancing of the different commercial efficiency factors so as to produce the maximum net return is the problem which confronts the modern engineer. To consider all the various commercial factors and to evaluate the resulting equation is something that never is perfectly done, but usually only moderately well done. It is not so hard to reason, given certain premises, but to make accurate observations and to draw particular conclusions, especially unpleasant conclusions, is difficult.

It is instructive to consider the conditions which have resulted in the formation of the three great electrochemical centers of the world. Considered in their basic relations, these are Niagara Falls, the coast of Norway and New York Harbor. Decidedly there must be unusual features that have existed to condense electrochemical plants around these places as nuclei. Norway has the cheapest power and is the most remote from the centers of civilization, though transportation by water is convenient and cheap, but the labor supply is rather poor. New York can produce power at a fair, but not low, cost, but is itself a great financial and industrial center and the supply of labor is abundant. Niagara Falls has fairly cheap power, midway between the cost in Norway and in New York, and lies between three great American manufacturing districts, with a good labor market. We can see the pretty balance of the commercial factors in the different characteristics of the products made in Norway, New York Harbor and Niagara Falls by Electra's fiat.

Norway makes principally nitrogen products—both nitrates and cyanamide—from atmospheri air. She needs cheap power

and air. Power is cheaper in Norway than anywhere else in the world. Air is cheap anywhere, hot in the tropics and from the promoter's lips, but no location has any monopolistic rights on air. Electrochemical plants locate in Norway when the cost of power is the greatest determining factor.

The electrochemical industry of New York Harbor—located in New Jersey as well as on Long Island—is entirely the electrolytic refining of copper bullion. The closeness of the brass and copper mills, the ease with which coal for reverberatory smelting is supplied, the dominance of American copper mines, the extent of the labor market, the Metropolitan banking facilities whereby loans on the valuable staple can be negotiated—all these factors have contributed many advantages for this special purpose, so that almost all Western copper smelters now ship their bullion East for refining and 75 per cent of the copper of the world is refined near New York.

At Niagara Falls power during the inception of the electrochemical industry from 1890 to 1900 was abundant and cheap (and while power is no longer abundant there for electrochemical work, it is still cheap). To this had to be added the good conditions for labor and supply of raw materials and the closeness to the potential markets for the then new products, such as aluminium, carborundum, graphite and calcium carbide. The optimist can see here why things as they are now have succeeded. Copper refining is impossible in Norway and nitrate or cyanamide manufacture from air is impossible in New York Harbor. Each is possible within certain limits at Niagara Falls, where the manufacture of nitrates from air has been a failure and the manufacture of cyanamide is a success.

Having in mind the bearing of commercial conditions on the location of these three electrochemical centers, we can see that in the United States we have widely varying possibilities for electrochemical industry. After the first period of industrial electrochemistry is over, characterized by the creation of new industries for the manufacture of hitherto unknown products, electrochemical activity has entered during the last year into a new era by taking hold of a wider and broader field. Electrochemical methods are being introduced into the older metallurgical and chemical industries. Steel refining in the electric furnace is the most momentous single feature of this evolution. But there loom up now at the horizon the latent potentialities of electrochemical methods—both wet and furnace methods—in non-ferrous metallurgy in the big and beautiful West, with its unlimited metallurgical possibilities; and it is no exaggeration to say that no one appreciates more clearly these potentialties of electrochemistry than the progressive Western metallurgists themselves. So this development is bound to come, and, in the nature of things, it must be profoundly affected by the availability of one or the other source of power.

Three sources of power are essentially available—water, gas and steam. The beginnings of electrochemistry were intimately connected with water-power developments. Besides Niagara Falls and Norway, this is true for Schaffhausen, for France (especially Savoy) and Italy, as well as for numeous smaller centers of electrochemical activity, of which only Sault Ste. Marie, Lockport, Penn Yan in this country may be mentioned; and it is noteworthy that the erection now going on of the first American works for the manufacture of nitrates from atmospheric air is intimately connected with the water-power de-

velopment of the Southern Power Company. Yet water-power has its limitations for electrochemical work in two directions. Firstly, if the specific nature of an electrochemical plant requires a location not too remote from an industrial center, such location means the nearness of existing industrial plants that can pay a higher price for power than an electrochemical plant can usually afford; if the electrochemist and the power salesman come together and the electrochemist offers half a cent per kw-hour (which may be the limit he can afford to pay) he is liable to turn the promoter of the hydroelectric concern irate. Secondly, the number of water-powers which can be developed very cheaply is limited. In many cases the engineering difficulties to be overcome in construction of dam and pipe line are so enormous and the capital investment becomes so high that, in spite of the relatively low operating cost, the rate charged for power must be high, as it is in at least one instance which we have in mind in the West. Nevertheless, it would be wrong to assume that such a water-power plant could not expect to sell power for electrochemical work. A water-power plant of this kind would not be built in a district in which a gas plant or a steam plant could supply power at less cost. And, on the other hand, if an electrochemical plant is erected in such a district, this is done in spite of high power cost, for the sake of the other advantages which the district may offer.

The erection of its own hydroelectric power plant does not seem good business for a new electrochemical enterprise in most cases, in view of the large capital investment that would be required. It is generally hard enough to get together the usually considerable capital which is required for the electrochemical installation itself proper. It is, therefore, better business, at least in the beginning, to buy the power from a water-power company. Water-power companies are usually reasonable in such circumstances.

From water-power we turn to gas. The gas engine is an attractive proposition, especially in a natural gas field or where blast furnaces or by-product coke ovens furnish cheap gas. The capital cost is much lower than that of a water-power plant and for a natural gas plant one-fourth cent per kw-hour may be considered a fairly representative figure. Indeed, were it not for the uncertainty about the life of the natural gas field, power generated from natural gas in a gas engine would be ideal for the electrochemical plant. The minimum figures for both capital cost and operating cost exist in this case. With blast-furnace gas or with by-product coke-oven gas both the capital cost and the operating cost are somewhat higher, but, on the other hand, the uncertainty is removed. Further, the gas will not be given away and some charge must be made for it. Probably the two most magnificent blast-furnace gas-power plants in existence are the two U. S. Steel Corporation plants, working in parallel, at Gary and South Chicago. At South Chicago the charge made to the electric furnace plant for steel refining by the Héroult process is half a cent per kw-hour. Of course, for electric steel refining the blast-furnace gas plant is logically the ideal solution of the power problem.

In many cases producer gas offers decided advantages for electrochemical work, as well as for chemical and metallurgical work in general. We have especially in mind a chemical manufacturing plant in Denver, which is conspicuous by the ingenuity and originality of its conception and methods. It shares with Standard Oil the distinction of turning absolutely everything into some useful product. Cheap bituminous slack is used in the gas producer and the cost of power is something like half a cent per kw-hour. But as the company refuses to be discussed in print, its plant is simply mentioned here to show that the advantages of producer gas for chemical work do not only exist on paper, but are a proved reality.

In turning finally to steam the advances made by the steam turbine in recent years are certainly striking. At the present time steam turbo-generators and accessories are sold very cheaply. The use of high speed and high pressure means that a maximum of $\frac{1}{2} m v^2$ is raced through a minimum of cast iron, steel and copper. Whereas the capital cost of a steam power plant was formerly $70 to $80 per kilowatt, the enforced competition among the companies building machinery for generating stations and especially the competition against the needs of the industries has reduced this figure about 25 per cent. Hard times make low prices. With coal of fair quality at $2 per ton f.o.b. plant and with plenty of condensing water, electrochemical power can be furnished by the turbo-generator very cheaply—so cheaply indeed as to make the average electrical engineer look skeptical.

But it must be remembered that the average electrical central station engineer does not appreciate the conditions of the electrochemical plant and its requirements. The usual electrochemical plant has a load-factor of over 95 per cent and also a high power-factor. Moreover, its land is cheap as compared with city land and its taxes low. Remembering all this, it is probably no exaggeration to estimate that with all proper charges power can be supplied under the conditions stated above at one-third of a cent per kw-hour, if the plant has a capacity of 5000 kw. Under the usual load conditions of central stations the cost will probably be more than twice this figure. There is necessarily a vast difference between a plant designed for and operating under a fluctuating load, as is the case with a central station, and a plant designed for a maximum of economy under a load that is fixed and constant, as is the case with the power plant for an electrochemical works.

While we have not discussed the possibilities of the steam engine for electrochemical work at all, it is but fair to say that this old reliable prime mover has been doing and is doing faithful service in numerous electrolytic copper refineries and will hardly be discarded here in the near future.

To sum up, the object of this discussion was to emphasize two points. Firstly, that the power cost is not all-important in an electrochemical process, as the different efficiencies must be balanced against each other. Secondly, that every kind of prime mover as its field in the electrochemical power plant. On the streets of New York City the push cart, the horse-drawn vehicle, the gasoline truck and the electric truck compete with each other in carrying goods. Each is used according to the needs and ability of the owner. So, in supplying electric power for electrolysis, electric smelting and refining, the steam engine, the hydraulic turbine, the gas engines and the steam turbine will continue to compete against each other. This freedom of choice in the power plant augurs well for the inevitable expansion of the electrochemical industry.

Chicago Meeting of the American Institute of Chemical Engineers.

The third semi-annual meeting of the American Institute of Chemical Engineers was held in Chicago from June 21 to June 24. All business sessions and meetings for reading papers were held at the Congress Hotel.

At the opening meeting on June 21 the members of the Institute were welcomed to Chicago by Mr. Wade, representing the Mayor of Chicago, and by Dr. T. B. Wagner, representing the Local Committee. Members were invited to visit the important chemical industries of Chicago, many of which were inspected during the meeting.

The reports of the various officers and committees were then presented. The membership committee reported a healthy growth, the present membership being 150. A very large number of requests for information with reference to conditions of membership are received, but on account of the high requirements for membership the number admitted is small.

The secretary reported that the third volume of Transactions would soon be issued. The growth in papers published is shown by the fact that the first volume contained 11 papers and 204 pages, the second 19 papers and 308 pages, while the third will contain 25 papers and about 410 pages. All papers are read by every member of the publication committee, which results in the rejection of papers of little value and the improvement of those published. A rapidly increasing demand is noticed for the Transactions.

The report of the treasurer showed a balance on hand of about $1,100.

The committee on medal reported $300 on hand and presented for approval or criticism a number of designs for the medal. The committee was opposed to the bust of a prominent chemical engineer on the medal or the use of the name of any chemical engineer for the medal, a symbolical devise being preferred. Non-members as well as members may compete for the medal which shall be known as the American Institute of Chemical Engineers' Medal, but the paper for which the award is made must be presented to the institute and published in the Transactions.

The report of the committee on chemical engineering education was read by the chairman, Dr. Samuel P. Sadtler. The most important part of this report consisted in the digest of the replies received to the questionaire on chemical engineering education sent out with the December Bulletin of the Institute. Thirty-seven replies were received to the 19 questions submitted. In reply to the question as to whether the prevailing system of chemical engineering education is adequate an emphatic "no" was given in a large majority of the answers. As the question evidently referred to a four-year course, the vote may be taken as approving the present tendency to lengthen the chemical engineering course to five years.

With reference to the subjects to be included in the course on chemical engineering a number of responses stated emphatically what seemed to be the general opinion that students should not waste time in college playing with "chemical practice," but that fundamentals should be thoroughly taught and experience with practical work acquired in the factory.

With reference to the defects of present courses the opinion seemed to be general that the college failed to devote sufficient time to those chemical manufacturing operations and appliances which constitute the element or operative units out of which nearly every chemical factory is built up.

A paper stating what could be accomplished in the four-year chemical engineering course was then read by Prof. J. H. James, of the Carnegie Technical School, of Pittsburg.

A paper giving a summary of the work of the committee on five-year engineering courses of the University of Ohio was read by Prof. James R. Withson, of the same university. The report of the committee and the two papers read on edu-

cation will be printed in the form of a bulletin so as to afford discussion by mail.

During the afternoon the plant of Swift & Company was inspected. Dr. William D. Richardson gave a brief explanation of the processes used after luncheon had been served by the company. The entire operation of killing, cutting, curing, smoking and cold storage as well as the soap plant and glycerine refinery were shown.

During the evening President Frerichs delivered an able address on "Methods of Attacking Chemical Industrial Processes." Complete detailed drawings and models of an installation for the manufacture of soda by the Solvay process were shown. The great difference in conditions and prices between Germany and the United States was shown by estimates on a process which would be a failure here, although very successful in Germany. A process for the extraction of bismuth from its ore was also given in detail.

On Thursday morning by a special car members reached Argo, Ill., for a visit to the large new plant of the Corn Products Refining Company. The very extensive use of concrete in this plant was noted and members were interested to learn how this material would withstand the action of sulphurous acid. All parts of the plant were inspected, including the converters, sugar and starch houses. When the plant is completed it will have a capacity of 100,000 bushels of corn per day. Many of the members spent the entire day at the plant. Others visited the laboratories of the Dearborn Drug & Chemical Works, after which an automobile sight-seeing trip around Chicago's beautiful boulevards and parks was taken.

On Thursday evening a subscription dinner was given at the Congress Hotel. The most interesting feature of the dinner was the presentation of the institute medal to Mr. F. J. Wood, of Marx & Rawolle, for the presentation of his paper on "The Multiple Effect Distillation of Glycerine," which was published in volume 2 of the Transactions of the institute. The process is remarkably economical in coal consumption, ease of control and operation as well as purity of product. It is in operation at the Brooklyn plant of Marx & Rawolle as well as in Toronto, Canada, and in England.

Dr. Edward Gudeman, of Chicago, acted as toastmaster and kept the company in the best of humor. The first speaker was President Frerichs, who called attention to the fact that the institute was organized just three years ago with a charter membership of 40. Mr. A. Bement, Dr. Geo. A. Rosengarten, Secretary Olsen and Prof. A. H. White also gave toasts.

On Friday morning the following papers were read:

"Industrial Chemical Calculations," by Prof. Jos. W. Richards. This paper will be printed in form of a bulletin and will come up for discussion at the next general meeting.

"Rapid Determination of Tin in Food Products," by Dr. Edward Gudeman.

"Distillation on the Continuous System," by Chas. L. Campbell. This paper gave a complete discussion of both the theory and practice of purification by distillation of complex liquids. Very fine samples of lead and silver linings on copper and steel were also shown by Mr. Campbell.

Friday afternoon the plant of the By-Products Coke Corporation at South Deering was visited, while some members visited the works of the Carter White Lead Company at West Pullman.

On Friday evening a paper by Dr. William M. Grosvenor on "The Institute and the United States Patent System" was read. In this paper the defects of the present patent system were shown. A communication from the St. Louis section of the American Society of Electrical Engineers was read on the same subject. It was urged that a special patent court should be established and that judges should be permitted to employ experts for independent advice. A committee was appointed to draft resolutions to be presented to President Taft urging reform of the patent system.

A paper on "The Practical Value of Calorific Tests on Anthracite Coal" and another on "Two Methods of Testing Asphalt," by S. F. Peckham, were read.

On Saturday the plant of the Indiana Steel Company at Gary, Ind., was inspected. An inspection car was provided to carry the visitors around this very extensive plant. Aside from its huge size the most noticeable feature was the clean and orderly appearance of the entire plant. The coke used is produced in Lemet-Solvay by-product ovens from which 80 tons of ammonium sulphate are produced daily. The motive power used is largely electrical, produced by 17 3600-kw generators driven by 34 gas engines using furnace gases. The open-hearth furnaces, of which there are 14, are heated by producer gas. Much interest was shown in the huge electrically driven rolling mill.

A portion of the members also visited the Standard Oil Refinery at Whiting, Ind., as well as the Universal Portland Cement Company at Buffington, Ind.

The attendance at the meeting was about 50 and marked interest was shown in all the sessions and excursions.

The Iron and Steel Market.

The moderate burst of market activity in June has been followed by a very dull July. The June activity was clearly due to the release of orders which had been piling up during the period of uncertainty in the minds of buyers as to whether prices would decline and which released by the reductions of May 29 indicating that price questions were settled for the time being. After this artificial freshet the stream of business dropped to the low level which is normal for midsummer. Rather strenuous efforts were put forth in the fore part of July by interested parties to conceal the relapse and make it appear that the market was still active. Incontrovertible evidence was furnished, however, that new buying was very light and that the mill activity, although reduced slightly from the June average, was still due in considerable part to business which had been booked in June. By the closing week in July mill activity had been further reduced, the average output of finished steel products by all interests being probably well below 60 per cent of total capacity, whereas in June a level of about 70 per cent had been maintained for a time.

The market has been quite colorless as to distinct developments. Buying has been light simply because to the reserve felt by buyers in a general period of conservative business action there is added the restraining influence of its being the summer season when many men are away on vacations and business is normally light.

The control of finished steel prices is substantially as firm as it has been at any time, though possibly not as secure as before the action of the Republic Iron & Steel Company in May, withdrawing from the counsels of the steel producers and pursuing an independent course as to the price of steel bars. This company is maintaining the same prices as other interests, but it has not formally returned to the fold, and with that example before the trade the possibility of one interest or another electing to pursue independent action, should such a course appeal to it, is somewhat increased. In no quarter can be seen any influence which would tend to advance prices, while the possibility of accident effecting reductions at one point or another, if remote, is not invisible. Should it at any time become apparent that a general reduction in prices would be likely to stimulate such a general buying movement as to fill up the mills and make possible an advancing market through individual action of producers, there is every reason to believe the course of declaring an "open market" would be pursued. This was probably the main influence prompting the break of February, 1909, for while that movement had the appearance of being an accident, time has given evidence that the course of the market, with a great dip

and subsequent recovery, was fairly well thought out in advance.

Only a small number of the mills closed July 1 for repairs and inventory. In some cases the only closing was for Tuesday, Independence Day, while in other cases Monday was taken as well and in fewer instances the whole week.

The wage scales of the Amalgamated Association have all been signed. These scales affect only certain iron, sheet and tin mills. Most of the iron mills of the Central West are governed by this iron scale, which was signed before the expiration of the old scale on June 30 by the Republic Iron & Steel Company and by the mills in the Western Bar Iron Association after a two days' conference at Cambridge Springs, July 6-7. The leading sheet and tin plate interest is entirely non-union, while about 40 per cent of the independent mills are organized. These agreed upon a continuance of the existing scale.

Little buying of wire products has followed the reduction of $2 a ton effective June 20. Existing contracts were adjusted to the new basis, and for a fortnight specifications on such contracts were slightly increased. It is between seasons for the wire trade, but the usual fall buying is expected to commence during the second half of August.

Pig Iron.

The drop of Southern iron to a basis of $10, Birmingham, which occurred in June did not bring out the buying for second half which was expected in many quarters. Early in July the directors of the leading cast-iron pipe interest, which frequently leads in the buying of Southern iron, met and concluded not to buy for the present. Other large consumers have shown little interest, and buying of Southern iron has continued from hand to mouth. In most cases of market swings Southern iron leads Northern by from two to four weeks, and Northern iron has also lagged, there being scarcely any forward buying, while here and there prices have softened slightly. A very interesting development has been the continued blowing-out of merchant furnaces in the Mahoning and Shenango Valleys until only six of the eighteen are in blast. Most of the furnaces which went out merely kept in blast until their supplies of ore were exhausted, while purchases were not made for the current season. If these furnaces do not take action soon they will pass into the winter without ore and an unprecedented situation will be revealed. Pig iron prices are regarded as below cost of production to these furnaces, unless they have ore supplies of their own, and can use the profit on ore to cover the loss on pig iron. A number of lake front merchant furnaces are idle, but not nearly as large a proportion as in the valleys. There has been further blowing-out of furnaces in Virginia and eastern Pennsylvania, and at last it is believed that production of merchant iron is below the consumption. The market at the close of July may be quoted as follows: F.o.b. Birmingham, $10 for No. 2 foundry; delivered Philadelphia, $15 for No. 2X foundry and $14.25 to $14.50 for basic; f.o.b. valleys, 90 cents higher delivered Pittsburg, Bessemer, $15; basic, $13; No. 2 foundry, $13.50; gray forge, $13; malleable, $13.25 to $13.50.

Steel.

Specifications for July deliveries of sheet bars against contracts have been very good, fully equaling those of June and exceeding those of May; billet specifications have been lighter than for June. There has been scarcely any buying in the open market. Billets continue quotable at $21 and sheet bars at $22, f.o.b. maker's mill, Pittsburgh or Youngstown, while with the June reduction of $2 a ton rods are quotable at $27, Pittsburgh.

Finished Products.

Prices of finished products, f.o.b. Pittsburgh, except where otherwise stated, are as follows:

Plates, 1.35 cents for tank quality, ¼ in. and heavier.

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Shapes, 1.35 cents for beams and channels, 3 in. to 15 in. inclusive, zees and angles 2 x 3 in. and larger.

Steel bars, 1.25 cents, base; iron bars, 1.25 cents to 1.30 cents, Pittsburgh; 1.27 cents to 1.30 cents, Philadelphia, and 1.20 cents to 1.25 cents, Chicago.

Wire nails, $1.70 per keg, base; plain wire, 1.50 cents; galvanized barb wire, 2 cents.

Black sheets, 28-gage, 2 cents; galvanized, 3 cents; blue annealed, 10 gage, 1.50 cents; painted corrugated roofing, $1.40 per square; galvanized, $2.55.

Tin plates, $3.70 per base box for 100-lb. cokes.

The Western Metallurgical Field.

Chemical Engineering at the University of Washington.

The growth of chemical engineering education in the West is typified by the work done in the University of Washington at Seattle. Special courses leading to the degree of Bachelor of Science in Chemical Engineering are open to students, under the direction of competent industrial chemists who are directing attention to the solution of some problems of the Northwest.

Referring to the course of study, the work in mathematics, physics and shopwork, is the same as that required for students in civil, mechanical and electrical engineering. The general engineering work taken by the student in chemical engineering includes mechanical drawing, surveying, mechanics and hydraulics from civil engineering; machine design, steam engineering and experimental engineering from mechanical engineering, and industrial electricity from electrical engineering. This work comprises 54 per cent of the chemical engineering course. Special emphasis is laid on the fundamental processes underlying industrial chemistry and the student is taught general inorganic chemistry, analytical, organic and physical chemistry, with supplementary courses in special lines of work. The study of raw materials used in manufacturing processes in the Northwest also receives attention and a special laboratory is provided in which industrial processes may be tried out on a suitably large scale.

This special laboratory is equipped with a steam still, steam superheater, continuous extraction apparatus, hydraulic press, filter press, alcohol still, wood pulp digester, steam jacketed vacuum still, vacuum dryer, clay testing furnaces, pulverizers, melting furnaces, rotary cement kiln, wood grinder paint mills, mixing machines, electrically heated wood distillation retort and cement testing apparatus.

The university invites the co-operation of local industrial concerns in solving industrial chemical problems. To this end the concern desiring any investigation to be made furnishes the means for prosecuting the work, and the university places its industrial laboratory at the disposal of a competent investigator. In this way it is hoped that some problems may be solved to the mutual advantage of the university and the community.

Dr. H. K. Benson, who is in charge of the work, has organized his students into a Society of Undergraduate Engineers for the purpose of better informing the student of the possibilities of chemical engineering, and giving him a broader outlook of the field before he enters commercial work.

First National Copper Smelter to Close.

An echo of the old controversy between the farmers and smelters of northern California is heard in the report that the First National Copper Company is to close its plant at Coram. This smelter, better known perhaps as the Balaklala, is one of several that were originally ordered by the court to make arrangements to abate the nuisance alleged by the farmers to arise from the free discharge of smoke and fume into the atmosphere. Pursuant to that decree the company installed, at considerable expense, what is considered to be the most efficient means known of condensing fume and reducing the sulphur content of the discharged gases. Perhaps it was too sanguine, and agreed to more rigid stipulations than were necessary. In any event, it appears that the company is unable to live up to its agreement, and the consequence is that the farmers have pressed their case to the extent of causing the company to cease operations.

The decision probably will have a more widespread effect than merely to close this one plant. Other smelters operating in the vicinity are working under an agreement to remove noxious solid and gaseous substances from their furnace gases, and the farmers will make tests to see that the provisions are being carried out. The First National Company undoubtedly will continue to make refinements in its methods in the hope that the plant can be operated some time in the future.

Bureau of Mines.

Dr. Joseph A. Holmes, director of the Bureau of Mines, was in Denver last month en route to Alaska to investigate conditions there for the Department of the Interior. While in Denver he was entertained by Colorado mining men who took advantage of the opportunity to present to him the mining situation in the state, and point out the possibilities for federal assistance through the Bureau of Mines. Perhaps Colorado is not so much in need of new mines as improved metallurgy, and it was on this theme that the speakers at two meetings dwelt at length. An effort was made to impress the Director of the Bureau with the fact that there were many problems to be solved in connection with the treatment of low grade ores, and that it would be to the advantage of the mining and metallurgical industry in general if these problems could be attacked by the Bureau. Federal assistance is sought because a long line of experimentation must be planned and pursued, and private enterprise is not always equal to the task.

Dr. Holmes explained that the Bureau was in full sympathy with the needs of metalliferous mining and the problems of ore treatment, but that there were no funds available with which to carry on such work. The metal mining interests will have to impress Congress with the need of this work if they expect to secure the help of the Bureau of Mines in solving their metallurgical troubles.

Improvement at Cananea.

In the annual report of L. D. Ricketts, president of the Cananea Consolidated Copper Company, some interesting details are given of the metallurgical work during 1910. The impounding of tailings and the recovery of the water is now accomplished in an inexpensive manner. The tailings are stacked in such a way as to allow the water to settle back of the dam. This water contains about 0.5 lb. copper per 1000 gal., and this metal is recovered by precipitation on iron before the water is re-used in the mill. The copper recovered repays the cost of impounding the tailings and in the future no more of these will go down the creek. The concentrator is making a saving of 75 per cent, which is exceedingly satisfactory.

In the reduction division the cost of treating a dry ton of new copper-bearing material was $2.69 as compared with $3.09 in 1909. Reverberatory costs increased slightly, but this will be changed when the extensive improvements inaugurated last year are in operation.

Tests of dust losses in the various departments showed that serious loss was suffered in the converter fume, amounting to about $30,000 a year. Dust chambers and a stack were accordingly constructed to handle these fumes, and were so proportioned that when all converters are running the velocity of the gas in the flues will not exceed 3 ft. per second. As a result of all the improvements made at the plant Dr. Ricketts predicts that the cost of treating a ton of new copper-bearing material in the reduction division will be not more than $2.25.

Drying in Vacuum.

By B. Viola, M.E.,
Member American Society Mechanical Engineers.

(The conclusion of the serial on evaporators and vacuum pans, published in the April, May, June and July issues.)

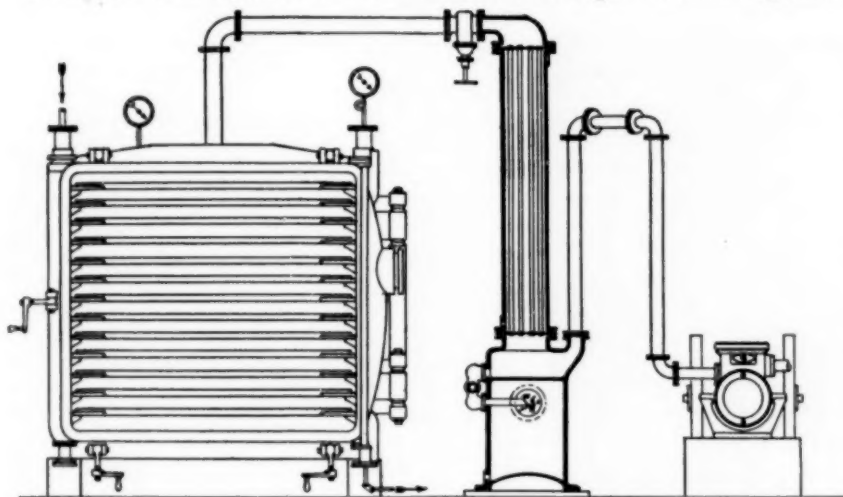If materials form crystals, as in the manufacture of sugar

FIG. 50.—ASSEMBLY OF THE PASSBURG VACUUM CHAMBER, DRYER AND AUXILIARIES.

or other crystallizable materials, we are able to concentrate the liquors till the crystals begin to form, let the whole cool off and separate the superfluous liquors from the crystals by some mechanical device.

It is, however, different when we want to get the dry substance of a solution which does not crystallize. Then all the dissolving liquor (solvent), whether it is water or some other liquid, has to be evaporated. To a certain density these solutions can be evaporated either by open fire or in a vacuum evaporator, yet in a great many cases the concentration does not suffice; in fact, when it is desirable to remove all dissolvents, it has been found impossible to do so by such an evaporation without a detrimental effect to the dry product, for the reason that such a high concentration causes an overheating of the material, and in this respect vacuum pans fail.

As the water contained in the solutions is reduced, the ebullition of the liquid becomes more sluggish and the temperature of the boiling point of the remaining water rises; consequently an overheating of the material when the final stage of concentration is reached, can very easily take place. The degree of concentration where such overheating would be injurious varies with different materials.

The concentration of liquids by the open or vacuum-pan method, effective as it is, fails more or less according to the material when the last traces of moisture have to be removed.

To supply the need thus created, and realizing the varied uses to which such an apparatus might be put, Messrs. Emil Passburg and W. Strohn, about thirty years ago, commenced investigations and conducted a series of practical experiments from which ultimately the invention of the vacuum-chamber dryers, Fig. 50, were brought about. This is the foundation of the present system of vacuum drying.

The apparatus consists of an evacuated iron chamber fitted with steam-heated shelves or horizontal coils, on which the drying material rests in comparatively thin layers on trays. The result obtained from these vacuum drying chambers was for many materials highly satisfactory, but at the same time a certain amount of labor was involved in charging the trays

with the wet material and again emptying them from the dry material. An apparatus which gave, if possible, still better results than the chambers, and at the same time insured a great saving of labor, was the outcome of the constructions, Figs. 51 and 52.

This apparatus, Fig. 51, which is being used in many industries with the greatest satisfaction, is known as the Passburg vacuum single-drum dryer. The apparatus consists of an iron casing of sufficient strength to withstand the outer atmospheric pressure when evacuated, which is very considerable, approaching about 15 lb. to the square inch, a steam-heated rotary drum which dips into the liquid and coats itself with a thin film.

On account of this very thin film of absolute evenness, which can only be obtained by dipping, the wet material is dried in one revolution, the number of revolutions, of course, being regulated according to the nature of the material, the percentage of water it contains, etc.

The thickness of the film of the rotary drum is regulated by the level of the solution contained in the trough or lower part of the outer casing of the apparatus and into which the drum dips. For this purpose observation glasses are provided, by which means the level of the liquid can easily be observed and regulated by a valve.

In the trough provision is also made for coils, through which the cold water passes in order to keep the liquid at the required temperature below the boiling point in the particular
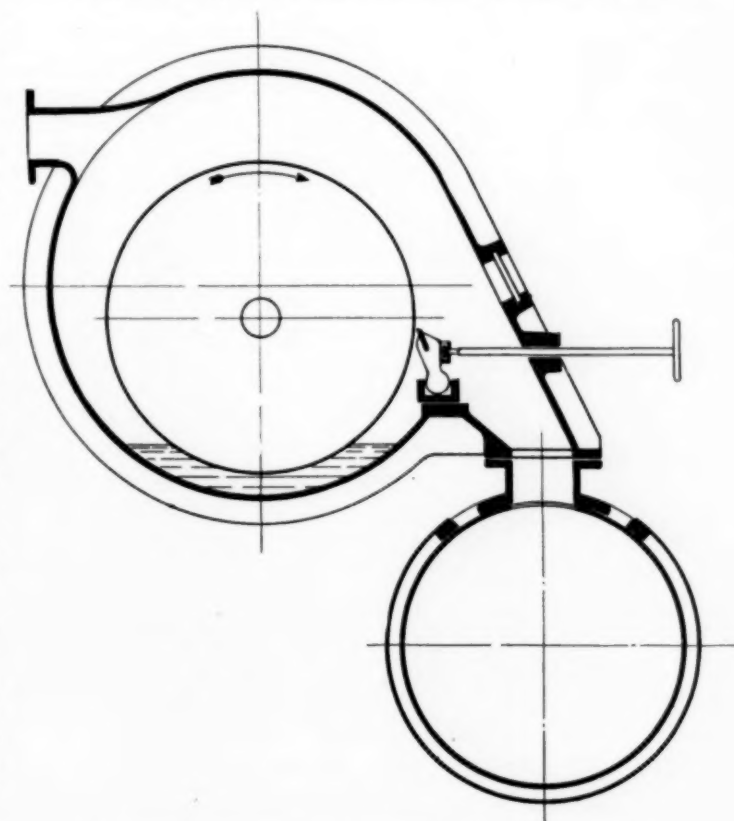

FIG. 51.—CROSS SECTION OF A PASSBURG SINGLE DRUM DRYER.

vacuum maintained in the apparatus. This feature is quite necessary, where milk, glue, and similar materials are being dried. Where, however, emulsions have to be treated, such as white lead, etc., and where a rapid precipitation of the solids

may occur, the place of the cooling coils is taken by suitably-designed agitators.

The ends of the rotary drums are protected by shields so that no liquid whatever may coat the former; in fact, it is only the cylindrical drying surface of the drum, which is at each revolution coated with and freed from the dried film and which comes into contact with the liquid to be dried.

The temperature to which the drying film is exposed rarely

FIG. 52.—ASSEMBLED PASSBURG VACUUM SINGLE DRUM DRYER.

exceeds 95 to 122° F. (35-50° C.), and as heating steam of from 20 to 25 lb. pressure may be used for heating the rotary drum, it will be apparent that the enormous difference be-

These drum driers work absolutely continuously and automatically, the liquid being drawn in by the vacuum maintained in the dryers and the dry material, which is removed by means of a scraper from the drum, drops into suitable receivers which can be arranged in such a manner that they can be emptied alternately and periodically without breaking the vacuum in the apparatus, and consequently without even temporarily interfering with the drying process.

Such a thin film of wet material in direct contact with the heating surface requires only a fraction of a minute to have all the water it contains evaporated therefrom by boiling; that is, by rapid and not by slow evaporation, and as this process takes place under a high vacuum, the temperature to which the drying material is subjected is consequently a very low one.

The number of revolutions of such an apparatus varies from 4 to 10 per minute, or in other words, the time during which the drying material is subjected to a low temperature can be counted by seconds.

Provision is also made by specially-designed apparatus, so that no loss of dust occurs.

The saving of steam, as no loss by radiation takes place, is naturally very great.

These apparatus offer the further advantage that where material of a poisonous nature, as, for instance, cyanide of potassium, white lead, etc., are dried therein, the laborers are fully protected, since by means of the automatic feed and discharge of such rotary-drum dryer neither the wet nor the dry material enters the drying room, and consequently all detrimental effect on the health of the laborers is absolutely prevented.

The author had the opportunity to work with such apparatus and to find the great advantage of the system, comparing it with the old-time hot-air drying system; that is, by slow evaporation. The most important fact is that the received dry material is always of the same quality, supposing a little attention is paid to the apparatus.
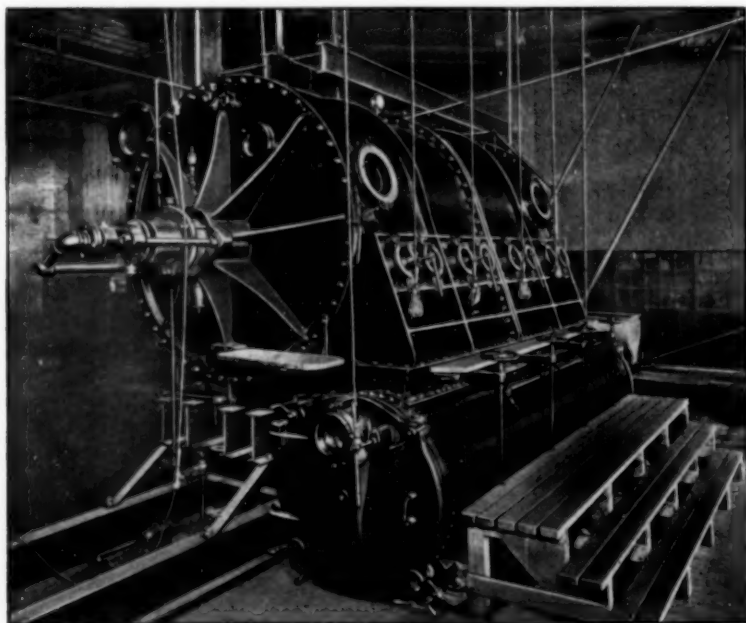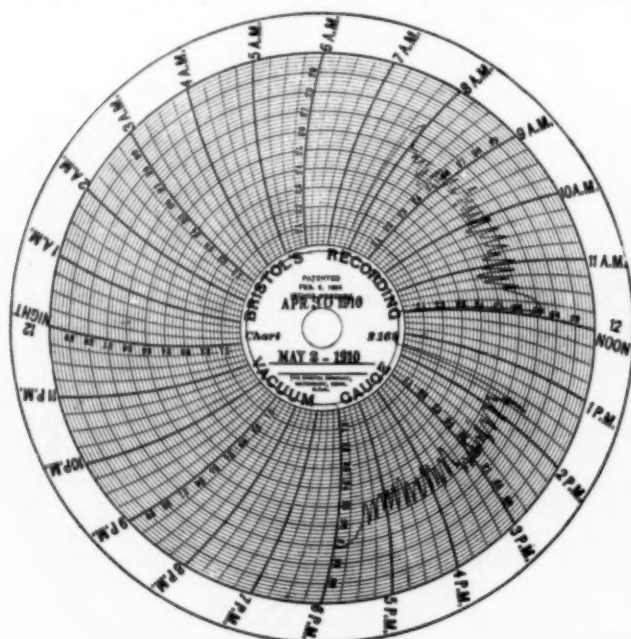
FIG. 53.—VACUUM RECORDING CHART OF A SINGLE DRUM VACUUM DRYER.

FIG. 54.—TEMPERATURE RECORDING CHART OF THE SURFACE CONDENSER CONNECTED WITH A SINGLE DRUM DRYER.

tween the temperature of the heating medium and of the boiling point of the water contained in the wet material, which in such a case as given above approaches 176° F. (80° C.), will insure a most rapid evaporation of the water, notwithstanding the low temperature at which such evaporation takes place.

The feed into the drum should always be carefully watched so that the liquid has approximately the same height in the receiver, and the heating surface of the heating drum be kept steady. Should the liquid in the receiver be used up to a certain extent, then it has to be fed again. This can be done either

by some automatic device so that the feed of the solution corresponds with the evaporating capacity of the drum, and if the former is once set for that purpose the liquor level within the dryer, into which the drum dips, will automatically remain the desired one, suitable for the material. Or it can be done by the attendant himself.

In this way little variations can be noticed on the vacuum, as
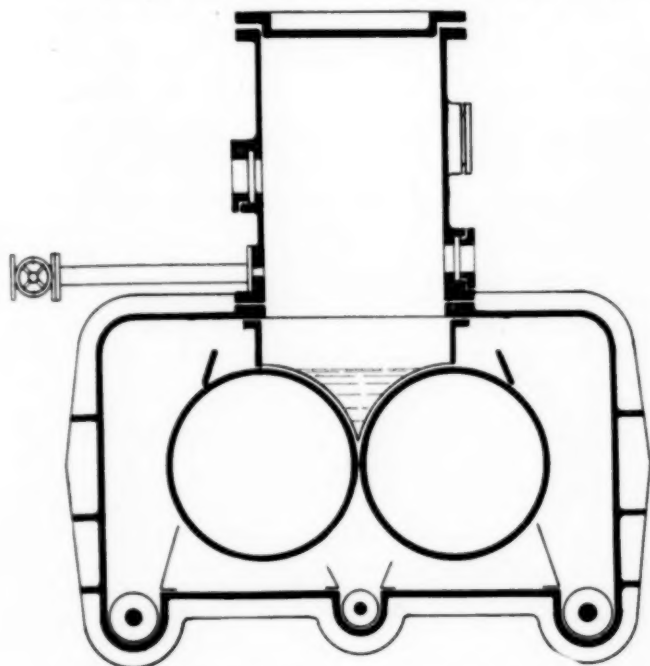
FIG. 55.—CROSS SECTION OF A PASSBURG VACUUM DOUBLE DRUM DRYER.

naturally, by feeding the drum with fresh liquid, a certain quantity of air is drawn in. These changes can be seen plainly on Fig. 53, the recording vacuum chart, and hardly amount to more than 1 in.

At what vacuum the drying process has to be performed depends upon the material, and it is an important fact that no two materials can be handled exactly alike. Every material must be treated on its own merits, and oftentimes small auxiliaries are very essential to the success of an entire apparatus.

Another important factor in vacuum drying, which affects eventually also the vacuum, is the temperature in the condenser. This should be kept approximately constant, and if this involves some difficulty it is the best to equip the plant with a temperature controller, to regulate the flow of the condensing water.

The chart of such a controller can be seen in Fig. 54, where the variations of this temperature are not more than 3° F., giving an excellent result.

The most remarkable difference can be seen on the chart at 7 a. m., when the recording thermometer shows about 81½° F., this being the temperature of the working room; but at that time the cold water was turned on, the drying began, and by and by was raised to the working temperature. At 11:45 a. m. the drying process was stopped again till 1:15 p. m., and this variation of the temperature can be seen.

Though the above-mentioned apparatus have proven most useful for drying liquids, still they could not be employed where materials of high concentration or pasty or similar form had to be dried, and this led to the invention and successful introduction of Passburg's vacuum double-drum dryer as illustrated in Figs. 55 and 56.

With this apparatus the wet material is either pumped or forced by some other means into the evacuated chamber,

within which two steam-heated drums rotate as shown. The material is fed on top of the two rotary drums, the latter by their rotary movement coating themselves by pressure with a thin film of the wet material, which is also dried in one revolution, similar to vacuum dryers containing a single drum only.

The thickness of the film on both drums is regulated by the space between them, which can easily be adjusted. Provision is made that any wet or insufficiently dry material which may drop from the drum is collected and again conveyed back to the feed pump, so that no loss in that respect occurs. The dry film is also removed from the drums by means of scrapers similar to the one employed in the single drum, and the dry material drops into suitable receivers, which again are emptied in a similar manner as described with single-drum dryers.

For materials which do not coat the rotary drum with an even film by means of the drum dripping into the material, a single-drum dryer with roller feed can be used, Fig. 57. This kind of dryer is used for heavy color pastes as produced in the manufacture of aniline dyes and similar materials. In fact, these pastes are so heavy that they cannot be drawn into the dryer by means of the vacuum maintained and have to be forced into the apparatus by means of the feed-pump.

From the construction or design of the above-mentioned vacuum shelf, single or double-drum dryers, it will have been noticed that no heat transmitted to such dryers has any means of escaping unused except by radiation from the outer shells

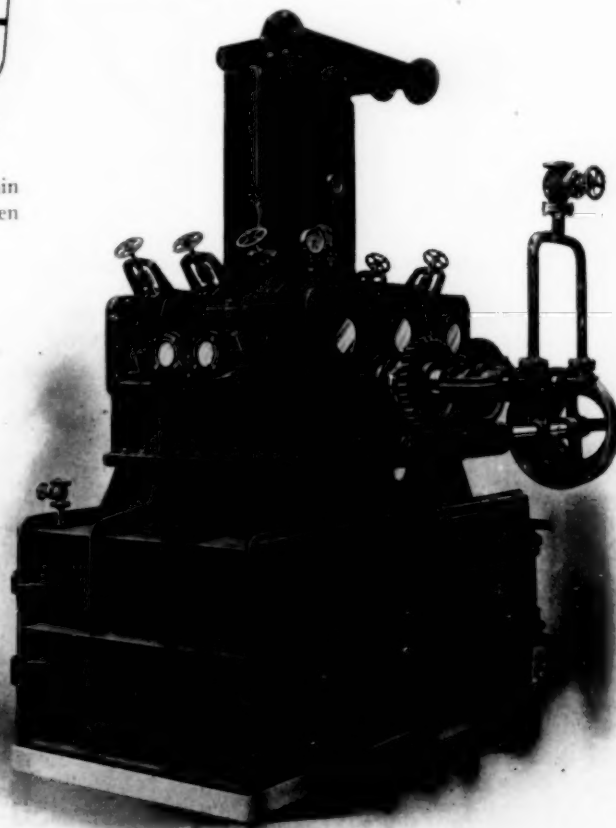FIG. 56.—ASSEMBLED PASSBURG VACUUM DOUBLE DRUM DRYER.

of the drying apparatus, and this can be minimized by an efficient coating of a good non-conductor of heat.

All other heat imparted by means of the heating medium, be it either steam or hot water, is absorbed by the water contained in the drying material for its own evaporation. There is no other outlet for any unused heat, as in the case of drying

under atmospheric conditions, where, as stated before, it is absolutely necessary to frequently exchange the air in the drying room, and in many cases heated air, in order to allow at least a fairly rapid evaporation.

As soon as the air in the drying room, be it hot or cold, has charged itself with a certain amount of moisture; that is, all that it can carry, it must be removed, as otherwise it will
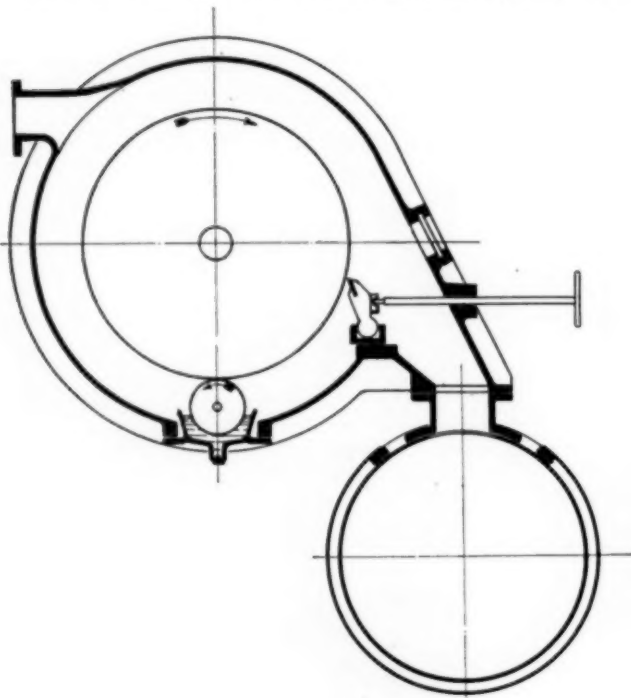

FIG. 57.—PASSBURG SINGLE VACUUM DRUM DRYER WITH ROLLER FEED EQUIPMENT.

directly stop a further evaporation of moisture from the material. This steam-saturated air, if remaining stagnant, forms a kind of fog or moisture-laden atmosphere in the drying room. A quick exchange of air is obtained by means of fans or similar apparatus, and the drying process is thereby aided.

With heated air an enormous quantity of heat is carried away by such air, runs to waste, and leaves the drying room unused.

The B. F. Sturtevant Company of Boston, Mass., states in one of its catalogs that even under favorable circumstances, where some of the heat was saved for reuse, the loss of heat by means of hot-air drying was as follows:

	Percent of total heat.
Required to vaporize moisture in stock	32
Lost by leakage, radiation from tower, and unaccounted for	37
Lost by radiation from return duct and by introduction of fresh air at fan room	31
Total heat imparted to air	100

The above data shows that only about one-third of the heat imparted to the drying air was really utilized for removing the moisture contained in the material. As this statement was made by a firm who are certainly experts and specialists in building hot-air drying systems, it may be considered as a fair statement, and certainly does not underrate the value of such drying systems. It gives an impartial proof of the unavoidable loss which is connected with any drying system where an air current is used.

Where materials of a hydroscopic nature have to be dried, absolute dryness can, of course, never be obtained by means of hot air, as naturally the drying material can never be dried below the moisture contained in the hot air used as a heating medium, and with materials which are of an inflammable nature at a certain temperature, hot-air dryers show great disadvantages, as they can easily produce an explosion, while with materials which easily oxidize the continuous heated air current has also naturally a detrimental effect.

Another important advantage of drying under vacuum is that the evaporation takes place in the absence of atmospheric air with all its impurities. Under such a rapid exchange as would be necessary when evaporating under atmospheric conditions, the air would partly deposit such impurities and thereby add to the quick deterioration of perishable materials, which deterioration may even continue in the dry material and thereby nullify the results of the drying or concentrating process.

Where discoloration is feared from the deposit of dust, the drying of such materials has no difficulty at all. When white lead, for instance, is dried by hot air under climatic conditions where the atmosphere is heavily laden with moisture, it is impossible to obtain either a dry product or a product with an even moisture content.

However, in a vacuum dryer the dry white lead has an even moisture content of .25 per cent, is free from a coating of dust or dirt, and has a white color.

Milk or egg powder, produced on the single or double-drum dryers, have proven to be practically free from obnoxious bacteria, lactic acid and overheated butter-fat, which latter tends to give the dry product a rancid taste.

On account of the low temperature employed during the rapid evaporation of the water in these liquids, which are partly emulsions and partly solutions, the albumen remained soluble also, and this is the most important and practically indispensable factor.

The result of the drying process is not only an increased keeping capacity of the dry products, as no germs are prevalent to start decomposition in the absence of water, but also the flavor of the re-dissolved powders could scarcely be distinguished from the fresh material.

This drying apparatus and process will consequently obviate the cold-storage system with all its recently much-discussed drawbacks and give the consumer a cheaply preserved and wholesome article of food at a reduced cost in all seasons of the year.

The old slow-drying method, which was only possible during the summer months when the atmosphere was not too heavily charged with moisture, has caused great loss to the manufacturers of glue; and even now, where a more rapid drying system by means of hot air and fans is frequently used, the gelatinized glue cakes either become mildewy and spoiled through the long drying process or the gelatinized glue becomes liquid again and flows through the net on which they were put for drying.

The losses have been obviated by the adoption of the vacuum drum dryers, which produce, quite independent of climatic conditions, a glue powder of even moisture contents, solubility and color.

I have briefly touched upon some of the various devices and types of apparatus used in evaporation and for the final removal of the last traces of moisture from materials which are desired to be thoroughly and perfectly dry in their final state.

The principles involved in their operation are many and complex. The removal of moisture from given materials is a scientific problem.

That its elimination cannot be successfully accomplished by "rule of thumb" methods is becoming recognized by those engaged in the manufacture of products in which drying is an important factor.

Brooklyn, N. Y.

The Application of the Turbo Blower in the Iron and Steel Industries.*

By J. E. Johnson, Jr.

I have been very much interested in the subject of this turbo blower since it was in its initial stages in the shops of the General Electric Company, the subject of blowing engines being one of profound importance to the blast-furnace manager. Men of this calling are commonly not members of the American Society of Mechanical Engineers and I trust that some discussion by one of the few who are will not be amiss.

The efficacy of the control of the volume of air blown is the question which will arise in the mind of every practical furnaceman as soon as he gives this subject consideration. Changes in quantity of air as small as 1 per cent or 2 per cent are frequently made in the quantity of blast delivered to the furnace as one of the means of controlling its operation, and this makes it obvious that exact control is a very necessary feature of any blast-furnace blowing apparatus.

I had the pleasure of seeing the installation of the turbo blower at the Port Henry plant of the Northern Iron Company about a year ago, and the simplicity of the governor is certainly admirable, but is a feature not exactly easy of comprehension to a man without thorough mechanical training, and a prejudice against these machines is likely to exist in the minds of many furnacemen on this account, which will make their introduction slower than it perhaps otherwise might be.

It is a well accepted fact that the efficiency of the turbine is far higher at low pressures than it is at high, while, on the other hand, it is equally well known that for high pressures only the reciprocating engine has a considerable advantage over the turbine. The result is that the most economical form of steam-driven apparatus is a combined unit in which the steam expands down to about atmospheric pressure in a good reciprocating engine and is then delivered to a turbine on its way to the condensers.

The very remarkable results obtained with this combination by Mr. Scott at the Interborough plant in New York have, in the opinion of many, given a decided check to the development of the gas engine.

In a similar way it has been known for years that for very low pressures the centrifugal blower, of which the turbo is simply a development, was more economical than a piston blower.

On the other hand, for higher pressures the piston compressor is undoubtedly to be preferred, particularly in blast-furnace work, on account of its operating as a meter as well as having much the highest efficiency as a compressor.

A condition of piston-blowing engine design is that the air cylinder must be large enough for the greatest volume to be handled and strong enough for the highest pressures attained, with the result of enormous size and very massive and expensive construction in modern blowing machinery.

In spite of their high cost it is notorious among operating men that really reliable blowing engines are exceedingly scarce. The problem of the valves is in itself one of great difficulty. It is almost impossible to get an air-inlet valve with sufficient area to allow the cylinder to fill absolutely without heavy loss by suction, and at the same time quick enough to give correct results at the speeds required of modern engines.

The turbo blower, on the other hand, suffers from the disability that it must have stages enough to blow the highest pressure ever required, although this high pressure will not, in ordinary operation, be required 1 per cent of the time, the normal pressure being perhaps only half as great.

If, therefore, we make a turbo blower to deliver air of a certain nearly constant, moderate pressure, and a blowing cylinder of comparatively small size, strong enough to deliver air to the

*A discussion of the paper of Mr. Richard H. Rice, presented at the recent convention of the American Society of Mechanical Engineers.

desired maximum pressure we shall have bettered the conditions of operation of both apparatus, increasing their efficiency and reducing their cost. In other words, the turbo blower and reciprocating compressor combined are more efficient than either apparatus alone, exactly as the engine-turbine combination is more economical in the consumption of steam.

For the best and cheapest blowing engine, therefore, we should have a combination of these two units, a turbine-driven turbo blower, delivering air partly compressed to the compressing end of a steam-driven blowing engine, the steam from the cylinder of this engine driving the turbine before passing to the condenser.

It may be objected that this is a complicated unit, but this is not the case. Most large modern furnaces require three of the best types of large reciprocating blowing engines to operate them. These are, in good modern usage, disconnected compounds—a high-pressure steam cylinder on one engine, a low-pressure on the next, etc.

A turbo-blower supplying air compressed to 8 lb. to two of these would enable them to deliver as much air as three drawing air from the atmosphere, and the efficiencies which are guaranteed in the use of exhaust steam by the builders, combined with the guaranteed efficiency of the turbo, indicate that this can be done at a decided saving in first cost and with an efficiency certainly no lower, probably higher, than that of either unit alone. This does not make any allowance for the possibilities of spray inter-cooling between the turbo and the reciprocating compressors, which, in ordinary practice, would be good for about 4 per cent additional saving at a very low cost.

The advantage of maintenance is undoubtedly with the turbo, as compared with reciprocating engines. At the same time the magnitude of the alternations of stress in the reciprocating engines is greatly reduced by delivering to them partly compressed air, and the problem presented by their inlet-valve gear, which has given so many bad hours to both furnacemen and engine designers, is practically eliminated by delivering to them denser air under pressure.

It, of course, cannot be denied for a moment that the efficiency of the reciprocating machine as a compressor is far higher than that of the turbo, except for the lowest pressures, while the advantages of positive metering of the air by the reciprocating engine will cause furnacemen to accept the combined apparatus without that hesitation which they feel toward the non-positive turbo.

The governing of such a combined unit is extremely simple. The ordinary governor on the engine unit does all that is necessary and no governor other than one to prevent racing is necessary on the turbine. The steam supply of the engine passes directly to the turbine. As long as all conditions remain constant the speed of the turbine will be unchanged, but if the pressure required by the furnace increases a little, more steam is admitted to the steam engine by its governor, and this increased quantity of steam speeds up the turbine to a slight extent and so causes it to deliver air compressed to a slightly higher pressure. This slight increase automatically compensates for any slight lag due to the greater load on the engine and for the lower volumetric efficiency of the piston-blowing engine at higher pressures as compared with low.

If, on the other hand, the speed of the piston engine is deliberately increased to deliver more wind, the increased supply of steam to the turbine will enable it to deliver a proportionately increased quantity of air at virtually the same pressure. Of course, some hand control will be applied to the turbine to enable the receiver pressures to be adjusted to the best conditions, but, once this is done, other changes will seldom be required.

For these reasons I believe that the steam-driven blowing plant of the future should consist of a turbine-turbo, piston-engine, piston-compressor combination.

This combination will have its greatest field in applications to existing plants where, as is so often the case, the existing blowing engine, while old, is by no means worthy of the scrap heap, but is unable to deliver quite the pressure or quite the volume demanded by modern conditions. In such a case the introduction of a turbine-turbo, taking steam from the engine and supplying precompressed air to it, will use what is now waste energy in the steam and apply it in such a way as to increase both the capacity and the permissible pressure of the existing blowing cylinder.

A further advantage which the turbo has as the primary machine is that of an inlet free from pulsations. These are much more serious in inlet than in outlet, for the reason that the volume and velocity of the air are so much greater.

One of the serious problems which had to be overcome in the application of the dry blast was the construction of cooling chambers whose walls would not be thrown down by the pulsations produced by the piston compressor on the incoming air. The smooth inlet of the turbo will eliminate entirely difficulties arising from this source.

The necessity of massiveness both in stationary and in reciprocating parts, and the conditions of operating the air valves, have operated to limit the best size of blowing engines, so that these are not only not increasing in size, but those now built are commonly as large as some that were built ten or twelve years ago.

The turbo-blower, on the other hand, lends itself admirably to enormous capacities in the same way that the turbine is commonly built for certain classes of service in sizes that have never been approached in reciprocating engines. For this reason it will not only be possible, but advisable, at large plants to install one turbo to supply several reciprocating engines, all of which, of course, will exhaust to it. This will contribute still more to low first cost, small space requirements and small maintenance charges.

I wish to express in conclusion my conviction that the introduction of the turbo compressor marks the beginning of a better era for the mechanical equipment of blast furnaces, but that its best and easiest application will be in conjunction with rather than in supplanting piston-blowing engines.

Ashland, Wis.

The Vacuum Filter Process in Western Australia.*

By T. B. Stevens and W. R. Degenhardt.

The stationary-frame vacuum-filter process was introduced into Western Australia four years ago, and has since been developed and improved until it now forms one of the chief methods of slime treatment on these fields.

The object of the present article is to point out, more particularly, the novel features of the plants in operation in this State.

Nature of Material Treated.—The material treated in vacuum-filter plants varies from the finest oxidized slime to an all-slimed product from sulphide ore. No slime of any grade has yet been tried that could not be economically handled.

The worst filtering slime yet tested was an oxidized clay containing no fine sand, obtained from the Never Never mine (near Southern Cross). This slime gave a cake only $\frac{3}{8}$ in. thick, containing 3.46 lb. of dry slime per square foot after 40 minutes of vacuum. It could be successfully washed and discharged. Average oxidized slime will form in 25 minutes a cake $1\frac{1}{4}$ in. thick, carrying 8 lb. of dry slime per square foot. An all-slimed sulphide product will, in the same time, form a cake $2\frac{1}{2}$ in. thick, carrying 18 lb. of dry slime per square foot.

It is always advisable to make experiments with a single filter frame before a plant is erected. The appearance of slime as regards its cake-forming power is often deceptive; thus, Rand slime is considered a perfect slime free from sand and is

practically a pure clay, but with this slime a 3-in. cake can be readily formed. The slime that contains most sand is not always the most easily formed into a cake. It appears that a well-sized slime, i.e., one containing particles of approximately equal sizes, best forms a cake, the reason being that the interstitial space through which the solution can pass is greater than in a mass of closely packed particles of variable sizes. This subject has been carefully studied in connection with sand leaching.[1]

The cakes from oxidized slime will, as a rule, contain 30 per cent to 33 per cent of moisture, while an all-slimed sulphide product will contain from 23 per cent to 25 per cent.

The Filter Frames.—The present filter frame is the result of considerable experimental work. The trouble with all the earlier frames was that the cakes cracked on exposure and consequently fell off. Practically all frames are now built of 1-in. steam pipe and are covered with 16-oz. duck, with cocoamat between.

The filter cloths are sewed with vertical seams 3 in. apart, this spacing having been found to be the best. Undoubtedly a
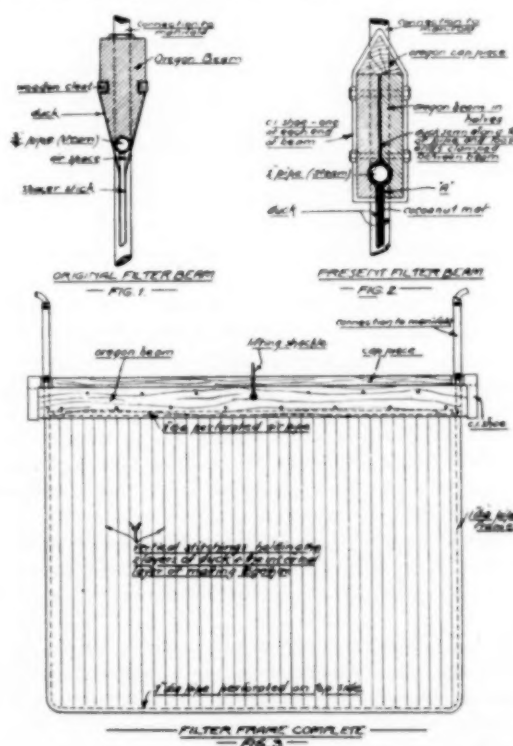
FIGS. 1, 2, 3.—FILTER BEAM AND FILTER FRAME.

great part of the filtered solution passes through the stitch holes instead of the cloth, and it has been found that bags with seams 6 in. apart do not take cake so quickly as those with 3-in. spacing. Bags with seams only 1 in. apart have been tried, but these did not form the cake so quickly as the 3-in., the reason being that the close stitching drew the cloth so tightly to and compressed the cocoa-mat that very little space was left in the mat for the solution to pass through.

The weight of the pipe, frame and cake is hung from two supporting beams of special section. The detail of these beams is important. The beam at first used (Fig. 1) had many disadvantages. The wooden cleats were not strong enough to hold the cloth and the weight of cake. The cloth also had no support where it joined the beam, thus causing cracks to form there on exposure. It is also obvious that if the cloth were allowed to stretch the cake would crack at the top.

Another difficulty was that the cake formed on the cloth where it was stretched over the beam and could not after-

[1] See "Cyaniding of Gold and Silver Ores." Julian and Smart; 1st edition; p. 49.

wards be dislodged by the air discharge, thus necessitating constant hand-cleanings. The air pressure also caused rapid wear of the cloths, for when the air was turned on a great strain was put on the vertical seams at the top of the bag, and it was only a matter of a few charges before holes appeared, allowing the slime to leak into the filtered solutions.

All these disadvantages have been overcome in the improved frame. (Fig. 2.) In this frame the support of the cloth is
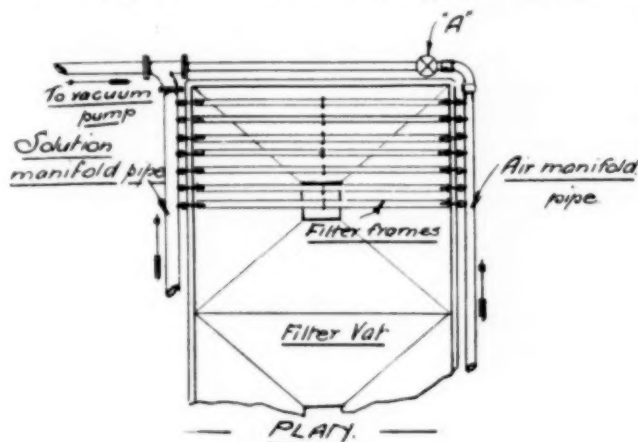

FIG. 4.—MODIFIED METHOD.

satisfactory, as it is clamped securely in the split beam. No cloth is exposed on the outside of the beam, consequently no cake can form on it. The shape of the edge A is designed so that it just presses on the cloth, but not so tightly as to prevent the passage of compressed air. The vertical seams come just up to this ledge, and it is, therefore, impossible for them to rip.

Frames are usually made of a uniform size (Fig. 3) of 9 ft. wide by 4 ft. 9 in. deep; this permits the use of a covering of duck 72 in. wide. The capacity of a plant is adjusted not by the size, but by the number of frames employed. Taking into account the handling of the frames for renewals, and the time occupied in filling and discharging the filter vats, this size of frame has been proved, from practical experience, to be the most efficient; deeper frames take considerably longer to discharge owing to the choking of the cakes.

If a plant is working on slime that will form a cake less than 1½ in. thick piping of smaller size than 1 in. may be used for building the frames, but with cakes above this weight nothing less than 1-in. steam pipe will stand the shock of the falling cakes. Pipes are liable to break where they join the beam, owing to the leverage exerted on them when the falling cakes spread the frames at the bottom. It is advisable not to fix the frames in the filter vat too securely, as a slight motion of the beams decreases the shock to the frames.

Various devices have been tried for obtaining the necessary spaces for the flow of the solution between the cloths of the frame. Wooden laths were first used, one being placed between each 3-in. seam; these were fairly satisfactory, but they did not give the support to the cloth that the cocoa-mat does and the cakes are, therefore, more liable to crack during exposure. Ripple iron, i.e., galvanized iron with small corrugations, as used at the Waihi mine, New Zealand, was also tried, but as this had to be used without vertical seams trouble was experienced in discharging the cakes. When the air was turned on the cake was stretched but not discharged.

With very free-filtering slime, i.e., slime that will form a cake of 2 in. or more in 20 minutes, it has been found that the cakes form thicker at the bottom than the top. The reason for this has been traced to the fact that the filter mat is not sufficiently porous to allow the solution to be drawn off as quickly as it passes through the cloth. This difficulty has been obviated by using the top pipe of the frame, as well as the bottom one,

for drawing off the solution. Fig. 4 illustrates this method, which may be described briefly as follows: In taking cake or washing, both the solution and air cocks on the frames, as well as the valve A, are left open, thus enabling each cock to draw from approximately half a frame. When the cakes are ready to discharge valve A and the air valves on the frames are closed; compressed air is then admitted to the other end of the air manifold and the air cocks are opened one at a time to discharge the cakes.

The same result could be obtained by using larger pipes for the frames and a larger solution manifold, but this would add needless weight to the frames. This arrangement is also useful as an addition to an old plant, where, by reason of the internal deposition of lime, the pipes become too small for the flow of solution.

Filter Vats.—The chief point to be considered in the design of filter vats is the method of discharge. It was obvious from the beginning that in Australia sufficient water could not be obtained to discharge them by sluicing with water pressure as practised in America. It was required to discharge the residue into the residue dam in the form of a pulp as thick as could be handled. This has been effected by mixing devices of different designs placed in the bottom of the filter vats.

Fig. 5 shows the most successful type, in which the cakes are dumped through discharge doors into a separate mixing agitator below. This allows of very quick discharge—less than 10 minutes—as the mixing can be completed after the discharge doors are shut and the filter vat is at work on the next charge. This type is always adopted where water from the Goldfields Water Supply is used for sluicing purposes, since water used exclusively for sluicing and not entering the treatment plant is supplied at a cheaper rate than that used for general purposes.[2]

Filling Filter Vats.—It is essential that the filter vats should be filled and emptied quickly. Two methods have been employed:

1. Filling by a small pump and gravitation. In this arrangement the pulp and wash are elevated to storage vats by small continuously running pumps, and the filter vats are rapidly filled from these through large pipes, as required. The excess pulp and wash from the filter vats is run out into two storage vats situated below and then elevated to the upper vats.

2. Filling by a large pump running intermittently. In this arrangement only one set of storage vats is required. These are placed on the same level as the filter vat, from and to which the pulp and wash are pumped as required.

Both methods have their advantages. The capital cost in each case is much the same, for, while with the gravitation plant extra storage vats are necessary, the pumping plant requires
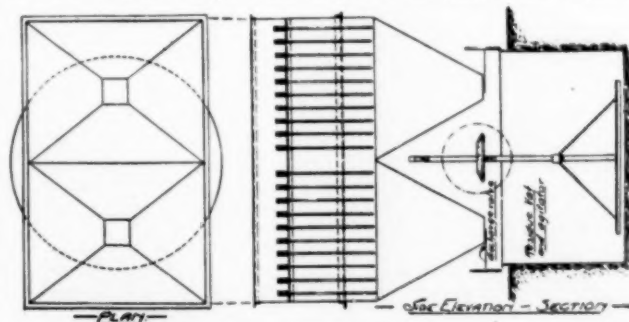
FIG. 5.—FILTER VAT.

a more powerful pump and larger piping and valves. As regards power cost, there appears to be little difference, since the gravitation plant requires power for running two small pumps continuously on a high lift, and for stirring two extra storage agitators, whereas the pumping plant requires only sufficient power to drive a large pump on a low lift.

[2] For general water services the price is 7s per 1000 gal. and for sluicing water is 6d per 1000 gal.

For a mine with a large power plant or one buying power from a central generating station, the sudden addition of a big load, such as a large pump, is of little account, but for a mine with a small power plant of its own the smaller but continuous load is the more convenient. As a rule, it has been found that where only one filter vat is required the gravitation filling is good, but where two or more filter vats are necessary the pumping plant is undoubtedly better.

Pumps.—For a high lift triplex plunger pumps are most efficient, and these call for no remarks. For a large capacity on a low lift the centrifugal pump is the best; the pumps most in use are of the Kershaw type. The casings of these pumps are made in halves with internal hard liners, which, with other working parts, are easily renewable. The cast-steel ported runner is fitted with cast-steel renewable shoes and is carried on a heavy mild-steel shaft. The shaft is run in a specially sealed four-part bearing with wedge-adjustable brasses, inside wearing ring and nipple. A stuffing box and gland for greasy packing are also provided. The runner shaft is connected by a muff coupling to a short shaft carrying the fast and loose pulleys between two outer standards having heavily flanged gunmetal bearings and loose thrust collars. All bearings are provided with water-circulating pipes and cocks to supply clean water under pressure to keep the bearings free from sand and grit. Stauffer lubricators are also fitted. The loose pulley is brass-bushed and fitted with a Stauffer lubricator; it is run on a cast-iron sleeve, which is secured to the spindle and which is the full width of the pulley face.

Wet-vacuum pumps without intermediate receiver have been found quite satisfactory and capable of maintaining a vacuum up to 27 in. These pumps are usually of 8½ in. diameter, single cylinder, 10-in. stroke, double-acting up to 40 r.p.m., according to the number of frames to which they are connected. They are fitted with vacuum and air vessels to promote uniform action. The pistons are of the trunk type, ordinary square packing being used.

Valves.—For filling and emptying the filter vats 6-in., 8-in. and 10-in. pipes are used. Where it is possible to do so quick-opening valves are arranged so that they may be operated from the working platform by means of extension spindles and hand-wheels.

In a plant lately designed 12-in. valves are proposed to be pneumatically operated. This improvement is suggested for the reason that most of the work in operating vacuum slime-plants consists of opening and shutting valves. Considerable power is required to open and shut a large valve, and even with a treble thread the manual operation is slow. The pneumatically operated valves act at once and will thus effect an appreciable economy of time and labor.

Clarifying Solutions.—In most cases it has been possible to make use of old filter presses for clarifying solutions; the solutions are, as a rule, very clear and a by-pass is arranged so that in the case of a leaky bag the delivery of the vacuum pump can be turned back into the filter vat. Where filter presses have not been available, filter-bag frames, placed in an elevated gold solution vat, form an inexpensive and effective clarifier. The frames are similar to those used in the filter vats, but are long and narrow. The suction is obtained by the fall to the zinc-boxes. The frames are usually cleaned once in each shift by closing the delivery to the extractor boxes and inflating the bags with air; this causes any deposited slime (which is very tough) to peel off into the bottom of the vat. After the frames have been cleaned the delivery is kept shut off as long as possible to allow the slime to settle. The settled slime in the bottom of the vat is sluiced out once a week into one of the treatment agitators.

Washing of the Cakes.—The time of economical washing varies greatly with the nature of the slime, and can only be definitely determined by taking progress assays from the discharge of the vacuum pump. This subject has been already discussed in an earlier number of the *Journal*. The time oc-

cupied in washing is always greater than that required for taking cake. It has been found that a low-grade slime (under 8s. per ton) obtains a good wash when the amount of solution sent to the extractor boxes is equal to about one-tenth more than that contained in the original pulp. As in all other filter processes, it is essential that the wash should be kept low in gold; to do this the filter vat must be quite free from pulp when the wash is put in and this can be easily effected by spraying down the sides of the vat with spent solution as the pulp is emptied out. If this be done, the wash will not gain more in gold by contact with the vat and cakes than can be counteracted by dilution with the make-up of barren solution and fresh water.

Notes on the Metallurgy of Wrought Copper.

By F. Johnson, M.Sc.

Metallic copper is unique among all the known metals by reason of its malleability, ductility, electrical and thermal conductivity, strength, toughness, comparative resistance to corrosion and immunity from oxidation, strength in torsion, capability of taking a high polish, high melting point, beautiful luster when chemically treated, ability to be worked into any conceivable shape, etc.

Its property of forging well at a red heat is one which enormously enhances its usefulness.

By reason of its great malleability, in which property it stands third in the table of metals, it can be hammered or rolled into sheets which have an immense range of application in the arts and in the engineering industries .

Its ductility enables it to be rolled and drawn into the finest wire, its possession of this property ranking it high in the order of ductile metals.

In electrical conductivity it is second only to silver, and this property alone has accounted for the production of millions of tons of the metal.

Its resistance to corrosion has enabled it to successfully withstand the advance of all competitors, as a material for such purposes as sheathing ships' bottoms or lining pump cylinders, while it also enables it to be used for culinary articles, chemical and distilling apparatus, boilers, etc.

Its immunity from oxidation is of great service wherever it is used, and this property, where nails, bolts or screws of the metal are used in contact with wood, gives it a peculiar advantage over iron. Unlike rust, what superficial oxide is formed has no corroding influence on the wood into which the metal enters.

Its thermal conductivity places it supreme among all competitors for use as firebox plates, stays and tubes of locomotive boilers, where its economical advantages over iron and steel are in the direction of greater durability and evaporative power.

Wherever heating surfaces are employed copper is preeminently desirable, its thermal conductivity being two and one-half times greater than that of iron. Steam, feed-water and many other pipes are also best made of copper.

Its toughness, tensile strength and torsional strength enable it to withstand all the stresses which it is ordinarily called upon or expected to encounter, e.g., telegraph and telephone wires and overhead electric trolley wires, etc. Any means of improving these properties, however, will always be acceptable.

Its luster, color and applicability to various shapes render it a favorite metal with art metal-workers; chasing, embossing and polishing being operations of ease to the skilled workman.

It is not proposed to say anything of the use of copper for alloying purposes, except that in all alloys which have to be rolled or drawn freedom from arsenic, lead and antimony is desirable. Copper is used for alloying with tin, zinc, aluminium and nickel chiefly, the number of variations of its alloys with these and other metals being almost infinite and their name legion.

The casting of copper, whether in the form of ingots for alloying, cakes for rolling into sheets, bars for drawing into wire, or billets for forging and drawing into tubes, etc., is an extremely important and delicate operation owing to the property which molten copper possesses of dissolving its own oxide. The elimination, in part, of oxygen is the keynote of copper refining, just as the regulation of the quantity of car-

FIG I.—STRUCTURE OF OXIDIZED COPPER BEFORE POLING, SHOWING DENDRITES OF CUPROUS OXIDE IN A MATRIX OF COPPER CUPROUS OXIDE EUTECTIC. MAGNIFICATION 100 DIAMETERS.

bon in iron is the keynote of steel making. Copper containing too much oxygen is cold-short, and, if oxygen is in great excess of the percentage contained in an alloy consisting entirely of the copper-cuprous oxide eutectic, the metal is also hot-short. If oxygen is removed to too great an extent, the metal then has greater solvent power for reducing gases, which it is incapable of retaining entirely during solidification, the expulsion of these gases causing the surface of an ingot of the metal to rise in the mold, thus showing a central ridge, or perhaps a worm-like excrescence, such a phenomenon being popularly known as "spewing" or "the worm coming out." (See Fig. 4.)
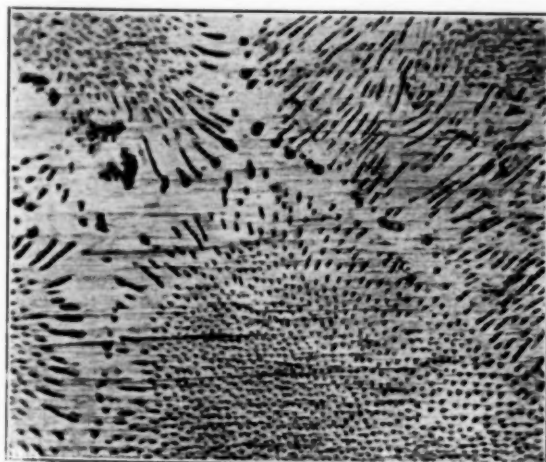

FIG. 2.—STRUCTURE OF COPPER AFTER ELIMINATION OF SOME OXYGEN BY POLING. SHOWS COPPER CUPROUS OXIDE EUTECTIC (0.47% OXYGEN). MAGNIFICATION 300 DIAMETERS.

All the phenomena occur with all kinds of copper when refined on a large scale, even with the purest electrolytic copper. When the oxygen is in excess the surface of an ingot shows a depression and it is said to be "under-pitch." When it shows an elevation, or "spews," it is "over-pitch"; when it exhibits a level surface it is said to be "tough-pitch," and in this condition the combination of its physical (including electrical) and

mechanical properties has reached its highest possible degree (for "refined" copper).

The foregoing are the broad general principles of refining, but every refiner of impure copper knows that it is possible to pole up charges of certain brands until an ingot will show a level surface, whereas the copper will be "red-short," and only by stopping poling at a stage where an ingot shows a depression can such copper be rendered fit for rolling or forging hot.

Micrographs 1, 2 and 3 show the structure of copper during the process of poling, illustrating clearly the reduction of cuprous oxide as poling proceeds. The cuprous oxide is completely soluble in the molten copper, but, as shown in the micrographs, is insoluble in the solid metal, separating out as a definite constituent, Cu_2O, which is light blue in color and is easily distinguished from cuprous sulphide by the property which it possesses of etching black when treated with hydrofluoric acid. Cuprous sulphide remains unchanged when similarly treated.

Where the Cu_2O exceeds 3.5 per cent to 4 per cent dendrites of Cu_2O make their appearance, as shown by the firtree-like constituent in No. 1. This is the first constituent to separate out during cooling of the liquid solution and its presence increases the viscosity of the copper before and during solidification, causing the copper to pour thickly. The other con-
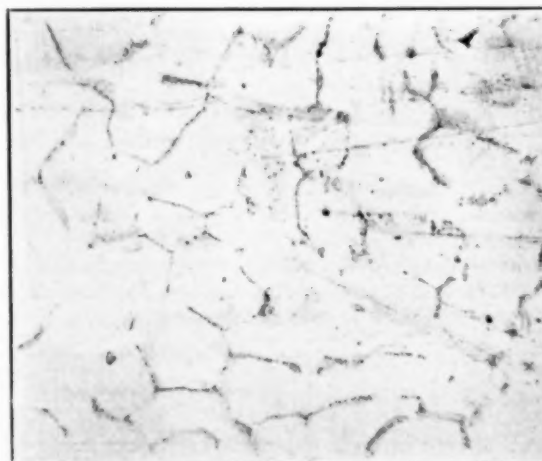
FIG. 3.—STRUCTURE OF TOUGH-PITCH COPPER, SHOWING CRYSTALS OF COPPER SURROUNDED BY EUTECTIC. MAGNIFICATION 300 DIAMETERS (0.08% OXYGEN).

stituent which forms the matrix is the eutectic alloy comprising an intimate mixture of minute copper and cuprous oxide crystals, or, as they may be conveniently called, "oxidules."

This eutectic structure is clearly seen in No. 2; its melting point and composition, according to Heyn,[1] are respectively 1084° C. and 3.5 per cent Cu_2O (see Fig. 5).

Fig. 3 shows the structure of "tough-pitch" copper, where, instead of the one constituent as in No. 2, there are two, viz., first, copper crystals which have separated out from the liquid solution, leaving the mother liquor (solution of cuprous oxide in copper) to feed them with copper until it reaches the eutectic composition, and, second, the solid eutectic itself.

Some interesting observations in connection with the properties of copper at various stages of the "pitch" have been made by Messrs. Hofman, Hayden and Hallowell in a paper read before the American Institute of Mining Engineers (Vol. XXXVIII). They prove that the gradual elimination of oxygen is synchronous with an improvement in electrical conductivity, but, strangely enough, the conductivity reaches a maximum while the copper is still decidedly "under-pitch" (0.2 per cent oxygen). They also prove that those traces of iron and sulphur which remain in "set" copper, after the oxidizing stage

[1] Mittheilungen aus den Königlichen Versuchsanstalten zu Berlin, 1910, XVIII, p. 315.

of refining, are not affected by the subsequent poling operation.

In a recent paper[2] the writer advanced a theory to account for the behavior of copper as influenced by the process of poling.

It must be remembered that when poling starts the bath of molten copper is very nearly saturated with cuprous oxide and
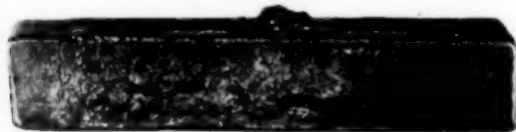
FIG. 4.—INGOT OF OVERPOLED COPPER, SHOWING "WORM."

that the effect of thrusting a pole of wood into such a bath is the rapid reduction of the oxide accompanied by the evolution of gaseous oxides of carbon, hydrogen, water-vapor, etc.

As the poling continues and the proportion of metallic copper increases, the latter has the property of dissolving some of the evolved gases, which it was formerly incapable of dissolving when saturated with oxide.

As more and more copper is thrown out of combination with oxygen, so more and more gases are dissolved. The gas for which copper has the greatest affinity under these conditions is hydrogen, which is probably derived from the decomposition by molten copper of the hydrocarbons in the pole, free carbon being precipitated and rising to the surface.

Trial ingots taken from time to time will show a depression at the surface and are said to be "under-pitch." This depression, however, becomes smaller and smaller until, when the charge is up to "pitch," it disappears and the surface of a trial ingot is quite level. If the poling be pushed beyond this point the surface will rise and form a convexity.

If sample ingots of (1) "under-poled," (2) "tough-poled" and (3) "over-poled" copper be broken and their fractures examined it will be seen in case (1) that the fracture is "dry,"
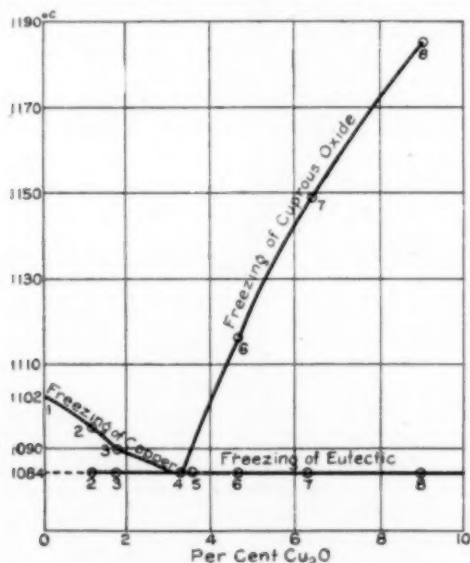
FIG. 5.—FREEZING-POINT CURVE OF ALLOYS OF COPPER AND CUPROUS OXIDE (HEYN).

with no silkiness of grain and no vesicular cavities; in case (2) the fracture is silky, with tiny vesicular cavities, uniform in size and distribution; in case (3) the fracture is still silky but there are more cavities, some of them larger than others, the larger ones occurring nearer the surface.

These cavities were evidently filled with gas, released at the moment of solidification; the bubbles of gas, by reason of

[2]Inst. of Metals, No. 2 (1910), Vol. IV.

their natural buoyancy, exerted an upthrust which tended to oppose the effect of liquid shrinkage and so prevented the surface from being depressed.

In the case of "under-poled" copper they were not sufficiently numerous to oppose this effect, while in the case of "over-poled" copper they occurred in such quantities as to not only oppose the formation of a depression, but to elevate the surface and in some cases to cause some of the semi-molten copper to be projected from the ingot.

In the case of "tough-poled" copper the bubbles of gas were present in just sufficient quantity to oppose the formation of a depression and no more.

Thus all ingots of comparatively pure copper, when poured at the proper "pitch," show a level surface.

The Effect of Mechanical Work on Copper.

Copper, after hot-forging, is stronger, more ductile and of closer grain than the cast metal. Hot-rolling can be carried to such an extent as to confer on the copper a tenacity which the same copper, after severe cold-working and annealing, cannot exceed. But hot-working should be stopped at such a temperature as will allow a certain amount of molecular rearrangement or recrystallization to take place following on

FIG. 6.—CAST COPPER DEOXIDIZED WITH MANGANESE, SHOWING "TWINNING." MAGNIFICATION 100 DIAMETERS.

distortion. That is to say, hot-wrought copper should have the closest and finest crystalline structure attainable short of the confused structure by working the metal at too low a temperature.

Subsequent cold-working may then be possible to a greater extent before annealing will be necessary. Cold-working, stage by stage, continuously strengthens, i.e., increases the tensile strength and limit of elasticity, but diminishes the extensibility. Le Chatelier states that pure copper can attain to a tenacity of 27 tons per square inch when drawn into wire, but this cannot be exceeded even by repeated drawing.. Annealing has the effect of lowering the tensile strength and raising the elongation.

These figures (for cold-wrought and annealed copper) have a limit, and the standard to take is that where maximum strength and ductility are just obtained in cold-wrought specimens which have been annealed for a known length of time at a known temperature in a non-reducing atmosphere.

Cold work produces a state of instability which may render the metal susceptible to "age-cracking" or secular brittleness.

There are two clever theories with regard to the cause of this state of instability. Firstly, however, let it be understood that the cold-working must be of sufficient severity to stress the metal beyond its elastic limit in order to produce deformation of a serious nature.

Beilby is responsible for the theory that those layers of the soft crystals which are contiguous to their boundaries assume a different condition—a hard, amorphous condition which confers on the metal, by virtue of its enveloping inter-cellular hardness, greater elasticity and tensile strength, greater brittleness and reduced toughness.

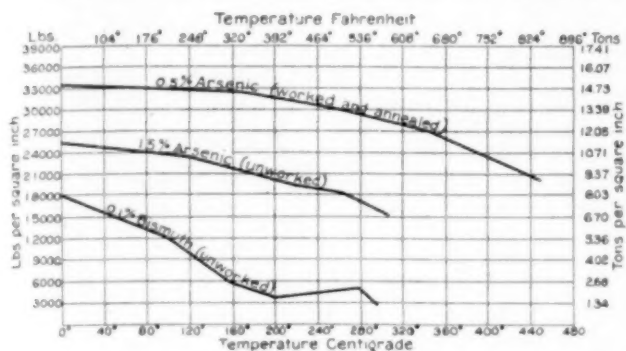In short, cold work distorts the crystals, causing gliding of

FIG. 7.—TENSILE STRENGTH AS FUNCTION OF TEMPERATURE.

crystalline elements over one another, along those planes where there is least cohesion; then appears the hard, non-crystalline variety which, thanks to its instability at a definite temperature, reverts back to the soft crystalline state once more when the annealing temperature is reached.

To Ewing and Rosenhain we are indebted for the theory of "slip-bands," and it will best serve the purpose of the writer to quote from an article written by Mr. Rosenhain for the *Times Engineering Supplement* of Nov. 6, 1907:

"It has been found that, when the application of force causes a crystal to change its shape, it accommodates itself to the new shape imposed upon it, by a process of sliding or slipping which takes place between the layers of minute crystalline elements of which each crystal is built up. After deformation the crystal remains a crystal as before, but the layers of its elements have been displaced relatively to each other. Further deformation results in further slipping, until, ultimately, the surfaces upon which slip can occur—originally plane in the undeformed crystal—have become converted into complicated systems of minute steps. If the attempt be made to push the deformation still further, then at some point or other of the cross-section of the metal two adjacent layers will slip far enough over each other to become entirely separated, and the fracture thus started will run through the metal and will follow all the minute steps of the surfaces upon which slip was at that moment taking place.

"The surface of the resulting fracture will thus be minutely stepped, and, in the case of a tensile fracture, the surfaces of these steps will all be sharply inclined to the direction of the extension. According to the degree of ductility of the metal and the scale of its original crystalline structure, the resulting fracture will show a more or less minutely stepped surface, giving the appearance of a fibrous fracture. That fractures so produced actually possess this stepped character can be clearly seen in properly prepared sections examined under high magnifications."

Mr. Rosenhain explained this theory somewhat fully at the second meeting of the Institute of Metals in January, 1909, during the discussion on "The Mechanism of Annealing of Certain Copper Alloys," by Messrs. Hudson and Bengough.

If metals are submitted to stress and at the same time their polished surfaces are viewed under the microscope the development of "slip-bands" may be clearly seen; they increase in number as the stress intensifies. Where the fracture ultimately occurs, parting takes place not along an irregular line following the boundaries of the crystals, but along the line coinciding with the least adhesive slip-bands or gliding-planes which have been developed through each crystal while submitted to stress.

Consequently where the structure is fine, the line of fracture will be straight and sharply defined; where coarse, it will be less regular and serrations will appear. So we may expect metal of perfectly homogeneous fine structure to be stronger than (a) metal of heterogeneous structure, i.e., where fine and coarse crystalline areas occur side by side, or (b) metal of homogeneous, but uniformly coarse structure.

In case (a) fracture will develop and rupture occur where the structure is coarse and the strength of the coarse areas will represent the strength of the whole metal.

In case (b) gliding-planes will be fewer in number but more pronounced, and their development less hindered by the hemming-in and arresting action of those less mobile inter-crystalline planes of greater mutual adhesion, which are more frequent where the structure is finer.

The effect of "twinning" is characteristic of wrought copper. Its explanation is perhaps rather obscure and is more easily understood by a study of crystallography. Suffice it to say that a crystal boundary of wrought copper may circumscribe one or more pairs of crystals (parallel laminæ), each pair having two sides coinciding, the other two sides running parallel thereto.

Distortion and recrystallization have completely disturbed the original polyhedral structure, the deformed and compressed crystalline elements having been compelled to take new and, as it were, unnatural situations. Even hot-wrought copper exhibits "twinning." Very homogeneous castings of copper or its alloys have a structure which is very susceptible to "twinning," the mildest deformation producing "twins." Micrograph (Fig. 6) shows a copper casting, deoxidation having been effected by means of manganese. The specimen was merely sawn from the casting, annealed for twenty minutes at 800° C. and slowly cooled, then polished and etched.

Behavior of Copper at High Temperatures

The influence of increased temperature on copper and its alloys, as on all metals, is to weaken the structure. Active crystalline growth, increased amplitude of molecular oscillation,
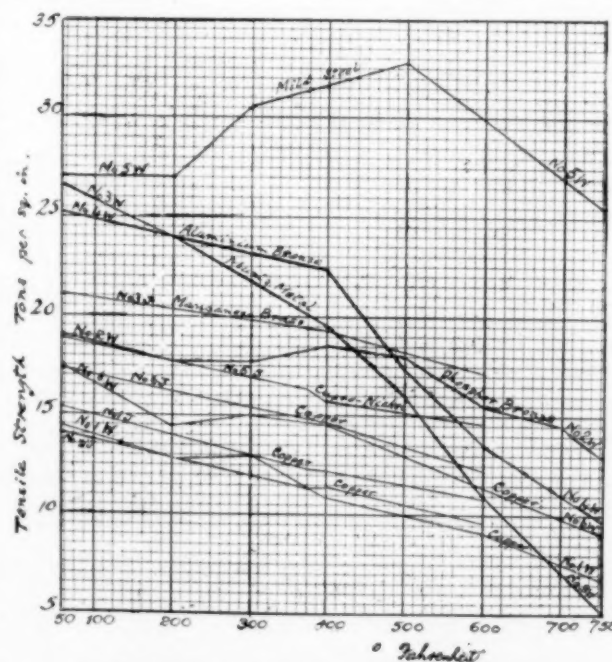
FIG. 8.—ALTERATION OF STRENGTH OF ALLOYS AT HIGH TEMPERATURES.

developing at still higher temperatures into incipient disintegration or, in the case of "burnt" metals into inter-crystalline oxidation, all render the metal less tenacious. Le Chatelier found that pure copper fell off in tenacity with increase of temperature from a tensile strength of 14.7 tons per square inch

at 15° C. to 12.9 tons at 200° C., and then fell off rapidly until at 530° C. the tensile strength was 4.6 tons per square inch.

Roberts-Austen [see plate 31 (reproduced in part herewith, Fig. 7) of the Second Report Alloys Research Committee[a]] showed that pure copper and arsenical copper showed little decrease in tensile strength compared with copper containing 0.10 per cent bismuth, which, at 160° C. or 320° Fahr., possesses only one-third of the tenacity which it possesses at 15° C. The copper containing bismuth was considerably weaker than the pure and arsenical forms, even at normal temperature. Although throwing considerable light upon the malignant influence of bismuth the results tend to give one an exaggerated idea of the potency of the evil for the following reasons:

Firstly, copper containing 0.10 per cent bismuth is rarely met with nowadays, and it may be said safely that the effect of

FIG. 9.—SHOWS ADVANTAGE OF COPPER OVER STEEL FOR FIREBOX STAYS, WHERE FIREBOX IS MADE OF COPPER.

bessemerizing is to remove bismuth from the pale of practical hurtfulness. Bismuth is very volatile, and Keller has shown that over 90 per cent of the bismuth contained in the original matte was volatized during the converting operation.

Gibbs states that "not only does the degree of elimination of any impurity depend upon the proportion originally present, but also upon the proportion of other impurities present." Be that as it may, bismuth is rarely met with in bessermerized
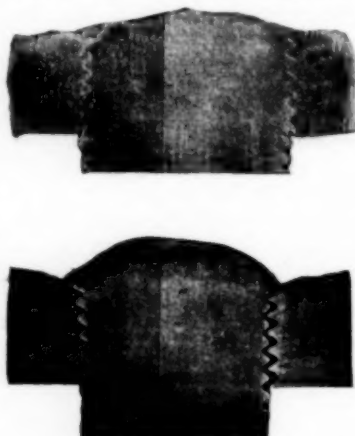
where electrical conductivity is of no account, and where toughness, strength and ductility are the desiderata, the presence of arsenic up to or even exceeding 0.5 per cent is eminently desirable, and this again suppresses the deleterious influence of bismuth, both at low and at high temperatures.

So it is important to know, having shown that Prof. Roberts-Austen's results are exaggerated by the artificiality of his otherwise splendid experiments, in what proportion bismuth is admissible in copper required by engineers to possess strength, toughness and ductility at all temperatures.

It is here that the writer would make rather full reference to a splendid paper read before the Institution of Civil Engineers in 1902 on "Locomotive Firebox Stays." The author, Mr. F. W. Webb, conducted a series of experiments on the suitability of various kinds of copper and copper alloys for staying locomotive fireboxes. The most frequent sources of trouble are those arising from *wastage of heads,* from *leakage* and from *breakages.*

(1) Wastage occurs principally in the vicinity of the brick arch where the heat is most fierce.

(2) Leakage occurs wherever the joint between the stay and the firebox plate has become loosened.

(3) Breakages occur principally at the top corners and may be attributed to the expansion and contraction of the inner copper casing. When they occur at the lower part of the box they are invariably due to overheating, consequent on becoming thickly coated with a non-conducting layer of boiler scale.

Exhaustive practical trials were made with engines fitted with stays of various kinds and extending over a number of years with the following results:

The above table has been compiled from the data furnished by Mr. Webb in the paper to which reference has been made.

In Fig. 8 are plotted the results of Webb's experiments on the various alloys used when tested at high temperatures. Their numbers and compositions are given in Table II, together with those of similar alloys tested by Mr. S. W. John-

TABLE I.

Material of Stay.	Total Mileage.	Mileage per Replaced Stay.	Description of Engine.	Boiler Pressure.	Cause of Replacement.
Copper...................	5,622,064	844	Compound Express Passenger	200 lb.	Wasted heads and leakage
Copper...................	277,800	277,800	Compound Express Passenger	200 "	Nil.
Munts Metal.............	169,058	262	Passenger	150 "	Wasted heads and leakage.
Brass { Cu 91 Zn 9	284,978	5,588	Coal-Engine	150 "	Wasted heads, leakage and breakages.
Bronze { Cu 97 Sn 3	113,657	791	Express Goods	150 "	Wasted heads and leakage.
Aluminium Bronze........	2,400	Not stated	Express Goods	150 "	Breakages.

copper in sufficient proportion to render the copper unfit for commercial purposes, although the writer has had personal experience of bessemerized copper containing sufficient bismuth to render it unfit for rolling hot.

As to the smaller quantities of copper produced by methods of roasting in reverberatory furnaces, including the processes of "bottoming" and "direct" refining, bismuth is not eliminated to nearly such a high degree as it is in the converter, while copper produced direct from blast-furnace smelting may possibly contain very nearly all the bismuth, arsenic, antimony, etc., originally present in the charge, so searching is the reducing action of the fuel and sulphur in contact with the metalliferous materials of the charge. Needless to say, very little of the world's supply of copper is produced direct from the blast furnace.

Secondly, commercial copper invariably contains an equilibrium-proportion of an element which profoundly modifies the harmful effect of bismuth, viz., oxygen.

Thirdly, in all copper required by engineers for purposes

[a]Inst. Mechanical Engineers, 1893

son under similar conditions. Webb's alloys are marked W, those of Johnson being marked J.

So it would appear that high tensile results at normal temperatures serve rather to indicate unreliability in practice. Nor do good tensile results at high temperatures necessarily indicate reliability in practice, e.g., mild steel. This only shows a mileage per stay of 610 as against 844 for copper, and this under much less severe conditions than the copper stays had to endure. The hardness of a metal when used for firebox stays where the fireboxes are of soft copper seems to be the main cause of failure. Screwing a hard metal into a soft one has the effect of abrading the thread in the latter, but, what is much more serious, subsequent riveting has the effect of expanding the hole in the copper firebox plate, causing imperfect contact between the stay and the firebox, and so leading to leakage. (See Fig. 9.)

Uniformity of tensile tests over a wide range of temperature is more probably a truer guide where the metal is of sufficient softness; the comparatively soft phosphor-bronze No. 2 W shows higher results and greater uniformity than copper Nos.

6 W, 1 W, 1 J and 2 J, but even its slightly higher degree of hardness leads to greater leakage. In this connection Mr. Webb's experience was that nothing provided so excellent a joint as a copper stay, the best joint being where the stay was of copper containing slightly less arsenic and therefore possibly softer than the copper plate into which it was screwed.

What is required is a metal as soft as or softer than commercial copper, of greater strength in tension, of greater elasticity, greater resistance to alternating stresses and capable of maintaining its superiority under the variable thermal conditions of a modern locomotive boiler working under high pressure. In addition it must have a melting point not appreciably lower than that of copper and a similar coefficient of expansion.

Cupro-nickel and cupro-manganese seem to be the most suitable alloys yet tried, although cupro-manganese does not wear so well as copper in the vicinity of the fire.

The greater tendency to red-shortness in all copper alloys

(b) The elongation for equal loads within the stretching limit increases generally with increasing temperature.

(c) The elongation is raised by cooling to — 20° C.; between + 20° and 300° C. it is nearly constant, while higher temperatures generally lower the elongation of the *annealed* material and increase that of the *mechanically hardened* material.

(d) The increased tenacity and elasticity conferred by mechanical working remain up to 200-300° C. practically unaltered, then, as the temperature increases, these properties fall off until at 500° C. they are the same as those possessed by the annealed material at that temperature.

The elongation of mechanically worked material remains lower than that of annealed material up to 300° C.; beyond this temperature it may have a higher or lower value up to 600° C., depending on its composition.

(e) As regards chemical composition it would appear (although this point is not clearly brought out in the research)

TABLE II.

ANALYSES OF MATERIALS TESTED AT HIGH TEMPERATURES (See Curves, Fig. 5.) (Webb & Johnson.)

No.	Description of Material.	Cu %	Zn %	Sn %	P %	As %	Ni %	Bi %	O %	Sb %	S %	Pb %	Mn %	Al %	Comb. Carbon %	Fe %
1W	Copper, deoxidized	99.76	nil	nil	0.01	0.20	0.04	nil	nil	trace	0.01	trace	nil	nil	nil	trace
2W	P. Bronze	97.07	....	2.91	0.02	....	....	....	....	....	....	....	....	....	....	....
3W	Muntz Metal	60.23	39.61	....	trace	0.10	....	....	....	0.01	....	trace	0.03	....	....	0.28
4W	Aluminium Bronze	92.91	....	....	....	....	....	....	....	....	....	....	....	7.13	....	....
5W	Mild Steel	....	....	....	0.07	0.04	....	....	....	....	0.03	0.48	....	....	0.13 Si 0.05	99.20
6W	Copper, deoxidized	99.89	....	....	0.08	0.01	0.02	....	....	....	trace	....	....	....	....	trace
1J	Arsenical Copper	99.18	....	trace	....	0.520	0.088	0.003	0.124	0.019	0.005	....	....	....	....	trace
2J	Arsenical Copper	99.42	....	....	....	0.27	0.04	0.012	0.108	0.013	....	0.06	....	0.008	....	0.022
3J	Manganese Brass	61.50	37.47	....	....	....	0.20	Mere trace	....	....	....	0.12	0.52	....	....	0.27
5J	Cupro-Nickel Alloy	95.51	....	....	....	....	2.91	....	....	....	....	....	0.02	....	Si 0.13	1.59
6J	Copper, deoxidized	99.84	....	....	0.103	0.007	0.16	0.001	0.009	0.005	....	....	....	....	....	....

TABLE III.

(FROM REPORTS FROM THE IMPERIAL TECHNICAL RESEARCH INST. IN BERLIN.) VOL. XVI., 1898.

Material P = Plate R = Rod	COMPOSITION %								ANNEALED MATERIAL.			
	Sn %	As %	Sb %	Bi %	Pb %	Zn %	Ni %	Fe %	Elastic Limit kg/sq. mm.	Tensile Strength kg-sq. mm.	Elastic Ratio	Elong. % on 60 mm.
P I	2.25	nil	trace	trace	nil	0.37	nil	0.03	8.0	29.9	27	58.2
R IX	1.86	nil	trace	trace	nil	0.43	0.04	0.10	6.5	28.6	23	55.1
P II	0.03	0.01	0.17	trace	0.02	nil	0.29	0.03	5.6	23.6	24	48.7
R X	trace	0.02	0.19	0.01	trace	nil	0.20	0.04	6.5	23.9	27	48.3
P VI	nil	nil	nil	trace	nil	nil	nil	0.01	4.9	21.9	22	47.7
R XI	0.02	0.04	nil	0.01	0.05	nil	0.15	0.04	4.8	23.8	20	51.0

N.B.—It is to be regretted that the oxygen contents were not included in the above figures.

of whatever composition is a serious defect, the breaking down of cold metal being more expensive than breaking down hot in course of manufacture.

Alluding once more to the testing of metals at high temperatures it should be noted that mechanical tests at the temperature of an air or oil bath are not comparable with the actual conditions in practice, e.g., where a stay is strongly heated at one end and kept cool at the other.

Mention must now be made of the work of the Royal Technical Research Commission of Berlin (1898) concerning the influence of heat, and of the conclusions at which that commission arrived after an exhaustive series of experiments on copper rods and strips of different composition and of different mechanical treatment.

Table III gives the composition of the different kinds of material used in the research and the following is a summary of the results of the investigation into the influence of heat thereon.

(a) The tenacity of the material is diminished with increase of temperature, the effect being much greater above 200°-300° C. than below.

that the more impure copper retains the proportionality derived by mechanical working better at higher temperatures and gains greater tenacity where the treatment has been the same.

It is very interesting to note that these experiments have been amply confirmed by Webb as regards the beneficial influence of tin. Copper containing 2 per cent or 3 per cent of tin, if carefully deoxidized, seems to behave better than pure or even ordinarily impure "refined" copper at high temperatures.

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A New Primary Cell.

In a paper presented before the British Institution of Electrical Engineers, Mr. W. R. COOPER describes a new primary cell invented by a Hungarian engineer, Stephan Benkő. As described, it is chemically an ordinary zinc-sodium bichromate cell and its original feature rests in the mechanical construction, which permits to overcome polarization at the carbon electrode. The fundamental idea is that polarization is due to the electrolytic change of the electrolyte at the carbon

electrode, and that, therefore, if the layer of electrolyte next to the carbon electrode is removed as soon as it is changed electrochemically the e.m.f. should be maintained constant and no polarization should set in.

The construction is indicated diagramatically in Fig. 1. Here the carbon is shown at C, and this is best described as a flattened cylinder, open at both ends. The hard surface of
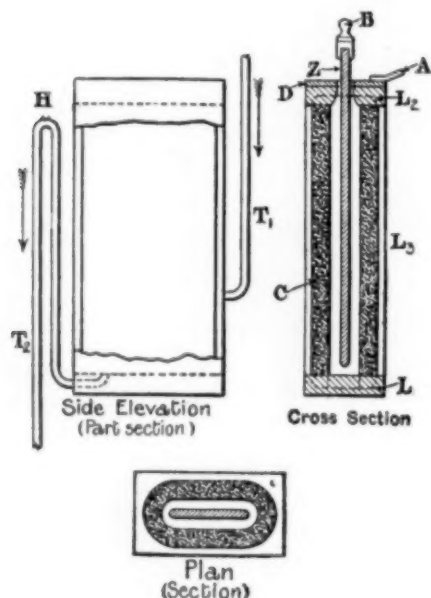

Side Elevation
(Part section)

Cross Section

Plan
(Section)

FIG. 1.—DIAGRAM OF BENKÖ CELL.

this carbon, as received from the carbon makers, is removed by scratch-brushing so as to leave it easily porous. The carbon is then provided with a lead cap, L_4, at the bottom, and a lead ring, L_6, at the top, thus providing a vessel open only at the top. A lead shell, L_5, of sheet lead 1 mm to 1½ mm in thickness, is then fitted round the carbon, leaving a small space between the carbon and the shell, and is jointed autogenously to the lead cap at the bottom and the lead ring at the top so as to form a chamber all around the carbon.

Thus there are two chambers, one inside the carbon and one outside. The outer chamber is provided with a Tube, T_1, and a second tube, T_2, is fitted to the lead base and is carried through to the inner chamber. The latter tube is carried up and bent over, and there is a small hole, H, at the top of the bend. Finally, a copper plate, D, corresponding with the lead ring, is soldered on to the lead, and forms one terminal, A, of the cell.*

The zinc, Z, is inserted into the inner chamber and carries the second terminal, B. When in action the electrolyte is delivered by the lead tube, T_1, into the outer chamber, when it percolates through the carbon to the zinc; it then flows off through the lead tube, T_2, the form of which maintains the level constant within the carbon. As there is a hole at H, syphoning does not occur, though this may be made to take place in cells so constructed by closing the hole with a finger for a few minutes. This is often convenient when it is desired to empty the cell.

It will be noticed that the current is carried away from the carbon at both top and bottom, and owing to this form of construction, and to the excellent contact between the carbon and the lead, very much heavier currents can be taken than is possible with the usual form of joint. A working drawing of what is termed the "Type A" cell is reproduced in Fig. 2, in which the lettering of Fig. 1 is retained as far as possible. The zinc plate is carried in an ebonite frame, F. The tubes T_1, T_2, are provided with unions and rubber washers, W_1,

*In later designs this copper plate has been abandoned, the terminal being jointed together to the lead ring.

W_2, for coupling to other tubes as may be found necessary.

The carbons at present are not made larger than is shown in Fig. 2, but if larger cells are required this is effected by fitting two or more carbons into one shell. These are thus permanently connected in parallel, and the zincs are correspondingly coupled together.

The particulars of the "single" cell are:

Width	1¾ in.
Length	6¼ in.
Height (including terminals)	9¼ in.
Weight (including zinc)	8.8 lb.
Maximum current continuously with L solution	17.5 amp
Power	26.25 watts

The current stated is for a voltage of 1.5 volts per cell, the e.m.f. being 2 volts.

In the case of a battery the electrolyte is contained in a lead-lined tank, from which it flows (due to its own head) down to a common supply pipe from which each of the cells is supplied by a tube, the junctions being by ebonite unions and rubber washers, as already described. Similarly the waste from each cell is carried into a common waste pipe, from which an inverted U-pipe is taken to maintain the level in the cells, and this discharges into a lower tank. By means of a cock the common waste pipe can be opened to this tank so that the cells can be drained when the supply of electrolyte has been cut off.

The arrangement is compact, the width across the front of the stand being only 25½ in. and the depth from front to back being 24½ in. In order to give the necessary head the over-all height is 7 ft. 9 in. for a 7-cell battery (and has recently been reduced). Air pressure might, of course, be used instead of a gravity feed; this would give greater compactness and might be preferable under certain conditions. But even with the arrangement shown, for a battery giving a minimum work-

Longitudinal section

Cross section

Plan

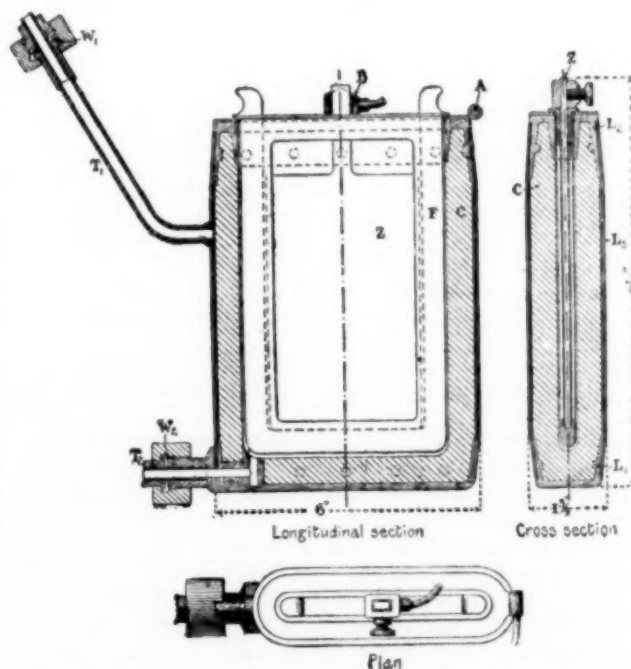FIG. 2.—STANDARD TYPE OF SINGLE BENKÖ CELL.

ing pressure of 10.5 volts and a power up to about 260 watts continuously, the space required must be regarded as relatively small.

With a given flow of electrolyte an extremely constant current can be maintained up to a certain limit. This limit depends largely, of course, upon the composition of the electrolyte. Various electrolytes may be used, and in the case

of bichromate solutions Mr. Benkö recommends the following formulæ: For electric light, 60 grams of sodium bichromate and 120 c.c. of sulphuric acid per liter of water (solution "L"); and for heavy currents, 100 grams of sodium bichromate and 150 c.c. of sulphuric acid per liter of water (solution "P").

The limiting currents obtainable continuously with the two first solutions in terms of the area of the zinc (taking both sides into the measurement) may be taken as follows:

	Per square decimeter	Per square foot
Solution "L" .... 6 amp		56 amp
Solution "P" .... 10 amp		93 amp

There is some local action due to the reaction of the acid electrolyte with the zinc, and this local action varies with the flow of electrolyte and with the current taken from the cell
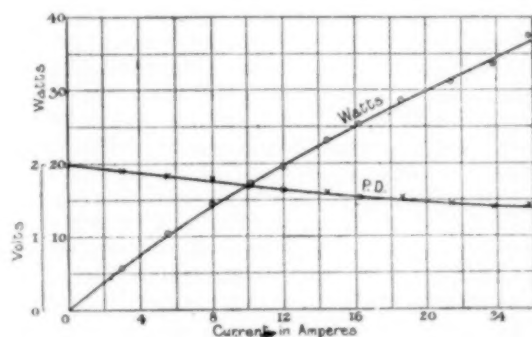

FIG. 3.—RELATION OF TERMINAL VOLTS AND TERMINAL WATTS
WITH AMPERES.

(owing to the fact that this alters the composition of the electrolyte) and with the state of the zinc.

There is also some polarization, but it is difficult to separate it from the internal resistance. Fig. 3 shows how the terminal pressure and terminal watts vary as the current is increased. The currents were not taken momentarily from the
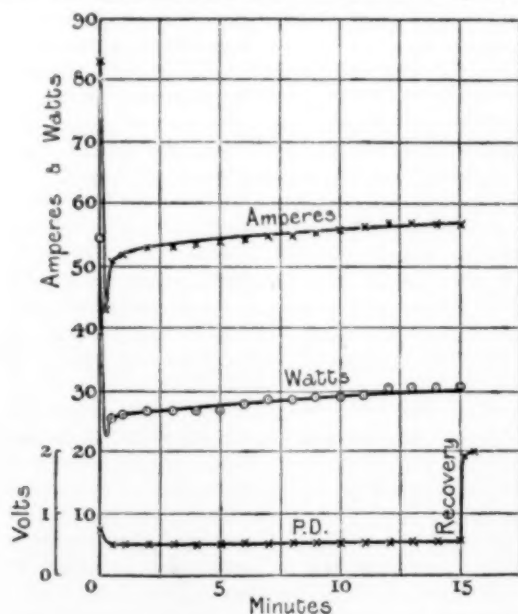

FIG. 4.—HEAVY CURRENT TEST ON CONSTANT RESISTANCE.

cell, but the external resistance was decreased step by step as quickly as the readings could be taken, and therefore polarization no doubt reduced the higher current readings to some extent. The highest outputs shown would not be maintained for any long periods, as the cell is overloaded. This test was made with solution L.

The test shown in Fig. 4 was made with a similar single cell, using the stronger solution P at a head (between the bottom of the tank and the top of the overflow tube) of 45¾ in., which gave a flow of about 1.9 liters per hour. In this test the external resistance was maintained constant, and the fall of current was observed for 15 minutes. The current might have been maintained longer, but the cell became very hot with such a heavy discharge.

Short-circuit tests with the L solution showed momentary currents of 100 amp at a terminal voltage of 0.5 to 0.6 volt, the current falling to 80 amp at the end of 5 seconds. With the stronger P solution momentary currents of 200 amp were taken at a terminal pressure of 0.5 to 0.55 volt. It follows, therefore, that the internal resistance is well below 0.01 ohm, an extremely low figure. This result is no doubt due largely to the excellent contact between the lead shell and the carbon, both at the top and at the bottom.

The Benkö cell has the advantage that it is small, easy to handle, and requires no careful attention. When it is not required it can be left on one side without fear of deterioration, the only desirable precaution being the running of some water through it to free it from the used electrolyte after heavy work. The laboratory, the charging of ignition cells, electroplating on a small scale, and possibly submarines (owing to the peculiar conditions involved) are fields in which such a battery would prove useful. There is also the electric lighting of yachts having no generating plant. It is also thought that the cell would be useful for telegraphy and in special cases for country-house lighting.

The simplest method of working such a battery is to turn on the flow of electrolyte when the light is required, and the flow would be turned off on retiring at night. With regard to labor, it may be mentioned that the zincs weigh 1 lb., and if we take 3 lb. (allowing for waste and unused zinc) per kw-hour, then a single cell will give ⅓ kw-hour and a battery of 17 cells will give 5 2/3 kw-hours before the zincs need be changed. If a house is using 1 to 2 kw-hours per day this means that the zincs are changed once or twice a week. As the operation is very simple, it takes very little time.

From the point of view of labor the electrolyte is more important than the zinc. With a constant output the amount of electrolyte per kw-hour will be, say, 7¾ gal. If, however, a battery such as we have assumed is allowed to flow steadily at a rate of ½ liter per cell, then a 5-hour run would require 9½ gal. This emphasizes the desirability of automatic regulation of the flow.

If hand regulation is adopted and not properly applied the amount of solution required might thus be as much as 15 gal. per day in the winter, or, say, 10 gals. per day if the electrolyte is used twice and "refreshed" by adding new electrolyte to it. On the other hand, with automatic regulation this quantity once or twice a week for 1 to 2 units per week would be enough. With proper arrangements the handling of these quantities would take very little time. These figures emphasize the fact that the cost of running depends largely on the method of running.

As far as can be judged the cost of repairs should be small or negligible, and equally the interest on capital is scarcely worth considering. Mr. Cooper concludes that electrical energy can be generated in this cell at about 50 cents per kw-hour under favorable conditions, and this cost will vary according to the carefulness of the regulation.

The Analysis of Manganese Bronze.*

By James R. Huber.

In nearly all of the specifications for manganese bronze castings it is quite noticeable that attention is called to the physical requirements and very little is said regarding the chemical analysis. This is no doubt partly due to a lack of knowl-

* A paper presented at the Pittsburgh meeting of the American Brass Founders' Association.

edge on the part of the engineers who draw up the specifications of the exact composition of this rather complex alloy.

A method for the analysis of this alloy was worked out in our laboratory last year and published in the METALLURGICAL AND CHEMICAL ENGINEERING for August, 1910 (vol. VIII, p. 463). This analysis presents several difficulties, especially the determination of zinc and aluminium. As we have gained knowledge by experience, we have been able to make some improvements on the original method.

A 5-gram sample is dissolved in 40 c.c. of 50% nitric acid (HNO_3). The small precipitate of metastannic acid obtained is invariably in such a finely divided state that much time is consumed in endeavoring to obtain a clear filtrate. A slight cloudiness in the filtrate will mean a result from 0.05% to 0.15% too low on the tin.

We find it advisable to cool the solution after boiling off the brown fumes, stir, and pour on a double S. & S. No. 589 blue ribbon filter paper. Fill the funnel once or twice and then apply suction, gradually at first, until the paper is well clogged with the precipitate. If the first of the filtrate is not clear, return it to the beaker and pour it on the paper again.

With these precautions, it is seldom necessary to repeat this operation and a practically clear filtrate is obtained. It is better to wash the precipitate with hot 10% nitric acid (HNO_3) instead of 2%, as the more dilute wash water has a tendency to start the precipitate running through the filter paper.

The filtrate is then evaporated to white fumes with concentrated sulphuric acid (H_2SO_4). Five c. centimeters is not enough, as the mass forms a hard cake and has a tendency to spatter when nearly dry. We now use 15 c.c. to 20 c.c. With this amount it will not be necessary to make a further addition when removing copper by electrolysis. The bronze will not contain any lead if made with high-grade spelter, but, should a trace be present, it is filtered off and weighed as lead sulphate ($PbSO_4$) on a platinum Gooch crucible. The filtrate is diluted to 500 c.c. in a 500-c.c. graduated flask.

Four 100-c.c. portions are measured out with a pipette or graduated flask into tall non-lipped 200-c.c. beakers. Two of them are neutralized with ammonium hydrate (NH_4OH), 7 c.c. of a mixture of 2 parts 50% nitric acid (HNO_3) and 1 part 50% sulphuric acid are then added and the solution diluted to about 150 c.c. The copper is plated out, using either a rotating anode or a solenoid. With the latter a brighter and more adherent deposit is obtained in less time than with the former.

The shape of the spiral anode causes the deposit to be more evenly distributed over the surface of the cathode and a much stronger current can be used. With a gauze cylinder cathode having 150 to 200 sq, cm of surface, 10 amp will deposit about 1¼ grams of copper in 15 minutes. Three amperes was the highest current we were able to use with the rotating anode and obtain a satisfactory deposit. We find it better to dry the cathodes in an air bath at about 100° C.

From these two portions we have duplicate determinations on the copper. In the filtrate from one the iron is precipitated with ammonium (NH_4OH), dissolved in dilute hydrochloric acid (HCl), reduced with stannous chloride ($SnCl_2$) and titrated with potassium permanganate ($KMnO_4$), according to Blair. In the other portion the manganese is determined by the well-known bismuthate method. Rinse off the anode with a little sulphurous acid (H_2SO_3) to remove the slight deposit of manganese dioxide which forms upon it and add these washings to the solution in the beaker. Bromine water and ammonium hydrate (NH_4OH) are then added, the solution boiled and the precipitate of manganese dioxide (MnO_2) filtered out and redissolved in sulphurous acid (H_2SO_3). Ten centimeters nitric acid (HNO_3) and 30 c.c. water (H_2O) are added, the solution boiled a minute and ¼ gram of sodium bismuthate added. Heat till the pink color entirely disappears. Add a few drops of sulphurous acid (H_2SO_3) to clear the solution, and boil again to expel all nitrous fumes. Cool the solution until quite cold and again add a slight excess of sodium

bismuthate. Stir a minute or two and then add 50 c.c. of 3% nitric acid (HNO_3) and filter through asbestos on a platinum cone, using suction. Wash with 50 c.c. more of 3% nitric acid (HNO_3), reduce with standard ferrous ammonium sulphate solution and titrate the excess with potassium permanganate ($KMnO_4$), which has been standardized against the ferrous ammonium sulphate.

The copper is removed from the other two 100-c.c. portions by electrolysis without the addition of nitric acid (HNO_3). The zinc is separated from the aluminium, manganese, and iron by precipitation with hydrogen sulphide (H_2S). Instead of reducing the iron by boiling with metallic iron or aluminium, or sodium thiosulphate, as suggested in Waring's modified method, we find it saves time to reduce with sulphurous acid (H_2SO_3). The use of aluminium is prohibited as we determine aluminium in the filtrate from the zinc. Methyl orange is used as an indicator when neutralizing the solution, and as it is not sensitive in a hot solution, it would be necessary to cool after boiling with any of the above reducing agents. In using sulphurous acid (H_2SO_3) it is only necessary to add about 10 c.c. to the cool solution, stir and neutralize with sodium hydrate ($NaOH$) and sodium bicarbonate ($NaHCO_3$), using methyl orange as an indicator.

We obtain the zinc sulphide in a granular condition and entirely free from iron by precipitating in the presence of hydrochloric acid (HCl) and ammonium sulphocyanide, as suggested by Kemmerer in an article in the *Journal of Ind. and Eng. Chem.* for September, 1910.

After neutralizing, dilute the solution to 400 c.c., add 4 or 5 drops of hydrochloric acid (HCl) and about 2 grams of ammonium sulphocyanide (NH_4SCN). Heat to boiling and pass a rapid stream of H_2S for about a half hour, keeping the beaker on a hot plate. Under these conditions the precipitate is rarely, if ever, slimy, but, should it form in that condition, it might as well be thrown away as it is impossible to filter it clear.

After the gas has been passing about 15 minutes, and the zinc is nearly all precipitated, add a drop of a dilute solution of sodium bicarbonate ($NaHCO_3$). If the black precipitate of ferrous sulphide (FeS) formed dissolves instantly, add another drop. But do not add any more if the black precipitate dissolves slowly. This precaution is taken to insure complete precipitation of the zinc.

There is a possibility that enough acid will be liberated in addition to the 4 or 5 drops added to prevent the last traces of zinc sulphide (ZnS) being precipitated. As this addition is not made until after most of the zinc sulphide (ZnS) is formed there is no danger of iron being carried down. Continue passing the gas about 15 minutes longer. Allow the zinc sulphide (ZnS) to settle, filter and wash with hot water. Dissolve in dilute hydrochloric acid (HCl) and precipitate as phosphate with sodium ammonium hydrogen phosphate ($NaNH_4HPO_4$).

The aluminium is determined in the filtrate after boiling off the hydrogen sulphide (H_2S) and concentrating to about 200 c.c. Add a little nitric acid (HNO_3) to decompose the ammonium sulphocyanide (NH_4CNS). Boil off the cyanogen fumes, add 5 grams of ammonium chloride (NH_4Cl) and precipitate the iron and aluminium as hydroxides by making the solution exactly neutral with ammonium hydrate (NH_4OH). Boil, filter and dissolve in dilute hydrochloric acid (HCl). Dilute to 400 c.c. Add 10 c.c. of a 10% solution of sodium ammonium hydrogen phosphate ($NaNH_4HPO_4$) and then ammonium hydrate (NH_4OH) until a slight but permanent precipitate forms. Then add 1½ c.c. of con. hydrochloric acid and 50 c.c. of a 20% solution of sodium thiosulphate ($Na_2S_2O_3$). Heat to boiling and add a mixture of 15 c.c. of a 20% solution of ammonium acetate and 8 c.c. of strong acetic acid. Boil 10 minutes, allow to settle, filter and wash with hot water. The precipitate is ignited and weighed as aluminium phosphate.

Lumen Bearing Company, Buffalo, N. Y.

Notes on Smelting in British Columbia.

By E. Jacobs.

Sinter Roasting at Trail.

The Dwight & Lloyd Metallurgical Company, of New York City, has put in two of its sintering machines at the smelting works of the Consolidated Mining & Smelting Co. of Canada, Ltd., at Trail, B. C.

There are a number of these plants in operation in various parts of the world, namely, nine in the United States, one in Mexico, five in Australia and several in Europe. The largest plant at present is one of those in the United States, with a capacity of 450 tons per diem. One still larger, of 500 tons capacity, is being erected at Broken Hill, N. S. W., Australia. Several others are under construction in the United States and Australia, respectively. Of those installed in the United States all but one have had, or are having, their capacity increased. The plant lately completed at Trail is the first to be installed in Canada, and has a daily capacity of about 200 tons dry weight fed to the machine.

The Dwight-Lloyd machine is particularly adapted to handling concentrates, flue dust and all classes of fine ores, converting them into a coarse cake which, owing to its strength and porosity, is highly suitable for reduction in the blast furnace. It will economically sinter and desulphurize mixtures having from 8 per cent to 18 per cent sulphur, and of sizes from ¼ in. down to 200-mesh or finer, making from them a coarse, porous product which will contain between 2.5 per cent and 5 per cent sulphur.

The machine proper consists of a structural steel framework carrying a suction box, a hopper, a line of cars or pallets, tracks for these to run on, and driving mechanism. The last mentioned consists of a pair of steel sprockets driven by a train of gears. The cars or pallets are planed on the bottoms and ends, so that they make an absolutely air-tight joint one with the other. In the bottom of these pallets are fastened cast-iron herring-bone grate bars, with slots ¼ in. wide. These pallets slide along the top of a large steel-plate suction box with planed cast-iron top. At each end of the suction box is a dead-plate which serves to make an air seal for the pallets as they pass on or off the suction box.

At Trail there are two large sprocket wheels about 8 ft. in diameter, on a heavy shaft, which constitute the means of operating the whole machine. On the underside of the machine there is a straight inclined track connecting at each end of the machine with a piece of curved track. The manner of operating the machine is as follows: The pallets are delivered by gravity into the sprocket wheel, the teeth of which engage the pallets by their small wheels or rollers, picking them up one pallet at a time. The curved track at the end of the machine serves as a guide to keep these wheels engaged with the teeth of the sprocket wheel. When they reach the top there are two guides on the end of a wind box on which the pallets come down, and the sprocket teeth then push them in a straight line over on to the wind box. At the same time they pass under a hopper full of ore which feeds a layer of ore mixture of uniform thickness onto the grate bars. Just at the point where a pallet passes onto the end of the wind box there is a gasoline burner by means of which the top part of the layer of ore is ignited. A high pressure exhauster is connected with the wind box by a pipe, and this draws air through the layer of ignited ore. By this means most of the sulphur contained in the ore is burned away, and at the same time the heat resulting from its combustion sinters the material, there being enough heat developed to slag the silica and iron contained in the mixture. While this is going on the pallet is slowly traveling across the wind box, and the speed of travel is so regulated that by the time it shall reach the end of the wind box the material is completely sintered. At the end of the wind box the wheels of the pallet are again engaged by the track which lifts the pallet and allows it to fall about 24 in. around the curved track at the end. It strikes the fallen pallet ahead of it with such force that the sintered cake is discharged from the grates and to a great extent cleans the slots of any adhering sinter. The cake then falls into a hopper or directly into steel cars with sufficient force to break it into pieces suitable for feeding into the blast furnace. The sintered product, therefore, does not require to be broken by hammers as is the case with some other processes. The length of the machines at Trail is 47 ft. over all. The effective hearth area is 42 in. by 264 in. There are 42 pallets on the machine, each 42 in. by 24 in.

The system of ignition depends largely on local conditions and the size of the plant. Gasoline is being used at Trail, but gas, crude oil, coal, coke, or even wood, may be used if more convenient. The gasoline burner is simply an arrangement which gasifies the gasoline in a coil made of brass tubing into the body of which the liquid fuel is sprayed by a needle valve. The gasoline comes out through a number of exceedingly small holes in the bottom of the body of the burner, and the flame is made to impinge against the surface of the moving layer of ore.

The supply of ore comes from large storage bins, one for each class of ore treated. From the bottoms of these bins the materials are fed automatically onto a conveyor belt, the proportion of each kind being regulated by varying the discharge opening in the bin. The conveyor belt discharges into a bucket elevator which, in turn, discharges to another belt conveyer placed near the roof of the building leading to a storage hopper above the sintering machines. By means of automatic feeders the ore is fed from this bin into a pug mill, and at this stage the required moisture is incorporated in the ore mixture, a special valve delivering the water in the shape of a fine spray. The mixed and moistened charge then falls into the hopper of the machine, whence it is fed onto the moving pallets.

The motive power for the whole system of mixing and feeding, and for the machines themselves, is simultaneously regulated, so that once the feeders are set to the proper openings for a given mixture the whole plant may be accelerated or slowed down at will by adjusting one speed regulator. This can be done without changing the proportions of the ore mixture, the composition of which will, therefore, remain uniform.

Any source of continuous power may be used. At Trail the power is supplied by three alternating current motors, two of which drive the fans and one the machines, feeders, elevators, etc. The total power consumption of this plant is about 75 hp, viz., 30 hp for each of the two exhaust fans and 15 hp for the remainder of the plant. The fans are specially constructed for heavy service, with water-jacketed bearings. They are run at a speed of 850 r.p.m. and produce a vacuum of 10 in. to 11 in. of water. The 24-in. pipes connecting the suction boxes with the fans are of riveted steel.

The Dwight-Lloyd machines are used for both lead and copper ores, both of which are smelted at Trail. The whole process, from the time the ore is dumped into the first receiving bins until the sinter is discharged into cars for removal to the blast furnaces, is absolutely automatic. At the Selby works, California, the sinter is also taken away from the machine by a steel bucket conveyer and automatically delivered to a 100-ton steel storage bin. The 200-ton plant at Trail is operated with only three men on a shift.

Mr. Henry J. Stehli, of New York City, superintendent of the Dwight & Lloyd Metallurgical Company, was at Trail in May, personally supervising the starting of the plant there. If successful, as it is confidently expected it will be, the operation of this plant will do away with the use of the Huntington-Heberlein roasting system now in use at Trail; also with the briquetting of flue dust, for this material will be mixed with the charge and sintered. The Huntington-Heberlein installation comprises seven roasters and 24 converters.

It is of interest to note, by the way, that the Dwight-Lloyd process is also being used for sintering fine iron ores and iron-furnace flue dust. A mixture of these is made and the heat developed by the combustion of the carbon in the flue dust is sufficient to sinter the entire mass into a solid but porous cake, which closely resembles that made from lead or copper ores.

It is claimed that three points of superiority are noted in comparing the Dwight-Lloyd process with other roasting systems, viz.:

1. The cost of installation per ton-day capacity of this plant is only one-half to one-quarter that of any other roasting appliance.

2. The cost of operation is from $1.50 to $2 per ton less than that of the old hand-rabbled reverberatory roasters.

3. In connection with several of the plants installed there has been a noticeable increase of the capacity of a blast furnace running on a charge containing sinter produced by this process; in one case as much as 20 per cent. This is due to the great porosity of the sinter and the fact that it is practically fused to a slag.

Trail Smelter Statistics.

The extent of smelting operations at Trail will be indicated by the following approximate figures, which are for the calendar year 1910: Tons smelted, 445,210; metal contents, gold, 138,901 oz.; silver, 2,017,007 oz.; lead, 33,871,837 lb.; copper, 5,282,139 lb. These figures for the company's fiscal year ended June 30, 1910, were generally higher, especially in lead production (42,368,816 lb.), while gross value of metals produced was nearly $400,000 greater ($5,911,767). Figures for nine months ended March 31, 1911, show a further decrease in tonnage and gross value of production, the tonnage smelted during this period being 304,587 tons, while the approximate gross value was $3,334,000, of which 55 per cent was gold.

The following figures give details of the company's operations for the nine months mentioned:

1910— Month.	Tons Received.	Tons Smelted.	Approximate. Gross Value.	Percentage of Gold.
July	35,641	35,138	$384,000	62.7
August	33,453	32,301	420,000	49.5
September	33,333	31,105	288,000	57.5
October	34,356	34,132	430,000	47.2
November	35,053	36,066	421,000	56.5
December	35,310	36,364	351,000	53.4
1911—				
January	34,505	32,958	349,000	57.7
February	32,323	32,315	326,000	57.8
March	35,096	34,208	365,000	54.8
Totals	309,070	304,587	$3,334,000	55.2

Of the total quantity of ore received, as above, 46,755 tons was custom ore and the remaining 262,315 tons came from the company's own mines, as under:

	Tons.
Center Star Group, Rossland	139,972
Snowshoe, Boundary	83,201
Sullivan, East Kootenay	25,362
St. Eugene, East Kootenay (concentrate)	6,129
Richmond-Eureka, Slocan	2,476
Queen Victoria, Nelson	1,984
No. 7, Boundary	1,718
Phoenix Amalgamated, Boundary	1,143
Molly Gibson, Nelson	330
Total output for nine months	262,315

The greater part of the ore from the company's mines was gold-copper, namely, 228,018 tons; the remaining 34,297 tons was lead-silver ore and concentrate.

Slag Disposal at Grand Forks.

The Granby Consolidated Mining, Smelting & Power Company, Ltd., operating a copper smeltery at Grand Forks, Boundary district, B. C., is preparing to dispose of its slag by a different system than that which has been in vogue for some years.

Up to the present time the hot slag has been hauled from the furnaces to the dump in slag cars by steam locomotives, but hereafter the slag will be granulated and disposed of in a manner described below.

The extent of the smelting operations of this company, the smeltery of which is stated to be the largest copper reduction works in the British empire, and second only to the great Washoe works at Anaconda, Mont., is indicated in the following brief outline of the smelting plant and equipment at Grand Forks.

There are in use eight rectangular, water-jacketed blast furnaces; six 44 in. by 266 in., and two 48 in. by 260 in. at the tuyeres; depth, 15 ft. 8 in., having a total capacity of 4000 tons to 4500 tons of ore per diem; 12 Connersville blowers, with a total air capacity of 239,000 cu. ft.; specially designed mechanical furnace charging system; hot slag haulage by 14-ton steam locomotives with 44-cu.-ft. slag cars; three converter stands 84 in. by 126 in., operated by electricity, and one stand 72 in. by 100 in. operated by hydraulic power; converter capacity, 50,000,000 lb. of copper per annum; three blowing engines with a total capacity of 19,700 cu. ft. of air per minute, and much accessory plant and machinery.

During the earlier years of the company's operations the slag was granulated by water and run to the dump by gravity in the usual manner. Then, as the dump filled to an extent, making it difficult to maintain the grade requisite for this method of disposing of the slag, it became necessary to substitute the hot slag haulage system which is now about to be dispensed with in favor of a return to water granulation and a method of disposal described below.

Four large receiving bins will be erected below the present level of the slag dump and at a distance of about 150 ft. from the blast furnace settlers. The slag will be granulated by water immediately after its discharge from the settlers, in the customary manner, and be conveyed thence to the receiving bins in the launders. From the bins it will be distributed over the dump by a series of conveyer belts, at first up a gradual incline to a maximum height of about 100 ft. and then by a cross-conveyer to the dumping place. These belts will be the ordinary 30-in. conveyers. The incline will at first be over a series of trestles, but later the grade of the dumped slag will be so regulated as to give the requisite incline up to whatever height it will be found economically practicable to carry it. The belts will be operated by two electric motors.

Two main reasons have caused the management of the company to make this change in the method of slag disposal. One is the excessive cost of the present haulage system, owing to the slag having to be hauled about three-quarters of a mile to the dump, and the high cost of maintenance of locomotives and slag cars. Second, the increasing difficulty of disposal by the present system, as the dumping-room below the present level becomes more and more curtailed. The substitution of the new method for that long in use is expected to involve an outlay of between $30,000 and $40,000, but important compensating results will be lower cost of slag disposal and provision of dumping-room for fully 10,000,000 tons of slag. To long continue the present hot dumping system would mean the changing of the course of the north fork of the Kettle River, with resultant cost of land that would be destroyed thereby, so the change determined upon will avoid a large expenditure in that direction and at the same time bring about a substantial reduction in operating costs.

It will probably be about four months, or near the end of September or beginning of October before the proposed change will be effected.

The Utilization of the Wastes of a Blast Furnace.

By Edward M. Hagar.

Until the last decade, practically the only utilization of the wastes or by-products of a blast furnace was the use of a portion of the waste gases to raise the temperature of the incoming blast through heating the brick work in so-called hot stoves, and in some cases a small portion of the power value of the gases was obtained by burning them under boilers to generate steam for driving the blowing engines.

At the present time the calorific value of the waste gases is being utilized directly in gas engines for blowing purposes and for generation of electric power, a considerable portion of the slag is used in the manufacture of Portland cement, and the flue dust, consisting of the finest ore and coke particles, is being collected and converted so as to be rechargeable into the furnaces.

The aggregate saving or profits resulting from these three developments is a matter of millions of dollars per annum, and in a modern blast-furnace plant it would almost seem that pig iron was the by-product, and, indeed, the investment in the equipment to utilize these former wastes exceeds that of the blast furnace itself.

The writer, in his work, has come in contact with these evolutions, with plants in operation or under construction, of a capacity to produce 12,000,000 bbl. of Portland cement per annum from slag and limestone, using over 1,300,000 tons of slag in a year, these plants being driven entirely by electric current generated by gas engines directly from the waste blast-furnace gases, the power requirements being 40,000 hp for 24 hours every working day.

In one of the cement plants the first commercial method for reclaiming flue dust was discovered.

By using the blast-furnace gases directly in combustion engines, after suitable washing to remove the grit, the power obtained from a given amount of gas is equal to at least two and one-half times that obtainable by burning the gas under the boilers for generating for use in steam engines.

A modern blast furnace of the usual size, with gas-blowing engines and gas engines driving electric generators, will provide sufficient gas to furnish 7000 kw of electric power, in addition to driving its own blowing engines.

This permits the most modern steel works, such as those at Gary, Ind., to practically do away with the use of coal for power purposes, operating the rolling mills by electric power from the surplus gases.

The United States Steel Corporation, of which the Universal Portland Cement Company is a subsidiary, has already installed 250,000-hp gas-blowing and gas-electric units, which, it can easily be figured, displaces or saves the consumption of approximately 1,000,000 tons of coal per annum as compared to the old-fashioned method.

With the modern high-blast pressures and the use of fine Missabe ore, the finest of the particles, together with the coke dust, are blown out through the top of the furnaces and are caught in the flues, dust catchers and gas washers.

The iron ore in this dust amounts to fully 3 per cent of the total ore charged, which aggregates the large amount of approximately 1,250,000 tons per annum in this country. Until within a few years this dust has been thrown away or used as filling, although containing about 40 per cent metallic iron.

For many years efforts were made to use this material by compressing it into briquets, but the cost of the operation, together with the fact that the briquets disintegrated and the dust was again blown out, led to an abandonment of the briquetting plants.

The first commercially successful method of utilizing the dust was discovered by passing the material through the cement

kilns at South Chicago. Experiments showed that with the proper heat treatment the coke dust could be burned off and the iron ore conglomerated into nodules or nuggets averaging over 60 per cent iron content. These nodules, when fed to the blast furnace, were easily and completely reduced. The fact that the sinter of the flue dust contains such a high percentage of iron and that all of the sinter is reduced, together with its physical shape assisting the steady movement of the charge downward in the blast furnace, thereby preventing so-called slips, makes the sinter more valuable per ton than any ore.

It was necessary to derive mechanical means for preventing the accumulation of the sinter on the walls of the kiln. Plants have been in operation for some years using this process, with endless chains carrying scrapers constantly passing forward through the kiln, and cooled in water on their return outside of the kiln.

Recently other methods of utilizing dust have been devised which may prove successful commercially, and the indications are that within a short time the greater portion of this former waste will be prevented.

The development of the Portland cement industry in this country and the extension of its uses have been marvellous, and the following table shows a remarkable increase in the production of Portland cement in the United States every year since 1895, when this country first reached the production of approximately 1,000,000 bbl.

Year.	Production of Portland Cement of United States. Barrels.	Production of Universal Portland Cement. Barrels.	Percentage of Universal to Total American Production of Portland Cement.
1895	990,324	....	....
1896	1,543,023	....	....
1897	2,677,775	....	....
1898	3,692,284	....	....
1899	5,652,266	....	....
1900	8,482,020	32,443	0.39 per cent.
1901	12,711,225	164,316	1.29 per cent.
1902	17,230,644	318,710	1.85 per cent.
1903	22,342,973	462,930	2.08 per cent.
1904	26,505,881	473,294	1.78 per cent.
1905	35,264,812	1,735,343	4.92 per cent.
1906	46,463,424	2,076,000	4.55 per cent.
1907	48,785,390	2,129,000	4.36 per cent.
1908	51,072,612	4,535,000	8.89 per cent.
1909	62,508,461	5,786,000	9.27 per cent.
1910	73,500,000 gov. est.	7,001,500	9.52 per cent.

It may be of interest to note the increasing percentage of the total American production shown by Universal Portland cement, which is the only Portland cement manufactured in this country using slag as one of the raw materials. With the new plant now approaching completion the aggregate production of Universal Portland cement in the Chicago and Pittsburgh districts will amount to over one-eighth of the country's total. Expressed in weight, the output of the finished product will be over 2,000,000 gross tons per annum.

Our plants in the Chicago district will consume all the available slag that is suitable for the purpose from an aggregate of nineteen blast furnaces in the South Chicago works of the Illinois Steel Company and in the Gary works of the Indiana Steel Company.

Comparing the pig-iron production and Portland cement production of this country in figures of long tons, the percentage of Portland cement to pig iron in 1890 was six-tenths of 1 per cent, in 1900 10 3-10 per cent, and in 1910 47 per cent. The continuation of any such relative growth would mean that before 1920 the tonnage of Portland cement would considerably exceed

that of pig iron. I would hesitate, however, to predict that such would be the case.

Portland cement is defined by the United States Government as the product obtained from the heating or calcining up to incipient fusion of intimate mixtures, either natural or artificial, of argillaceous with calcareous substances, the calcined product to contain at least one and seven-tenths times as much of lime, by weight, as of the materials which give the lime its hydraulic properties, and to be finely pulverized after said calcination, and thereafter additions or substitutions for the purpose only of regulating certain properties of technical importance to be allowable, not exceeding 2 per cent of the calcined product.

From this definition it will be seen that the raw material for Portland cement is not limited to any particular form of material. It may be made from any combination of materials that together furnish the proper elements. In this country Portland cement is manufactured from a number of raw materials, which, with a few exceptions, may be classed under four heads:

First.—Argillaceous limestone (cement rock) and pure limestone.

Second.—Clay or shale and limestone.

Third.—Clay or shale and marl.

Fourth.—Slag and limestone.

In all cases the raw mixture is a combination of some form of clay and some form of lime, and in the first and fourth classifications the clay materials contain some lime. This simply reduces the proportion of lime material necessary for a proper mixture.

In the manufacture of Portland cement from slag and limestone the molten slag flowing from the furnaces is granulated by a stream of water, loaded into cars and transported to the cement plant, where it is dried in rotary driers and receives the first grinding; it is then mixed in automatic weighing machines with the proper proportion of ground and dried calcite limestone. These are then ground together and burnt to a hard clinker at a temperature of nearly 3000° Fahr. in rotary kilns, using pulverized coal for fuel.

This clinker, after seasoning, is crushed and ground and mixed with a small percentage of gypsum to regulate the setting time. The cement is ground to such fineness that 96 per cent passes through a sieve having 10,000 meshes, and 80 per cent passes a sieve with 40,000 meshes to the square inch. It is then conveyed to the stock house for storage prior to shipment.

It is necessary to use a flux in furnaces supplying slag for cement manufacture, a pure calcite limestone. The limestone burnt with the slag must also be a pure calcite stone. It is also essential that the ores be of a uniform and proper character.

Inasmuch as Lake Superior ores are noted for their remarkable uniformity of analysis, the resultant slag obtained from the use of these ores and a pure calcite limestone is more uniform in its analysis than any form of natural clay deposits used in the manufacture of Portland cement, and the variation in the proportions of the two raw materials used in the manufacture of Portland cement from slag is less than those of any other materials mentioned above.

In addition, the opportunity for analysis and selection of the proper ingredients through the use of an artificial material is a great advantage as compared to the necessitous use of natural materials just as they are found with their variations in analysis at different depths.

In the intense heat of the kiln, under the influence of the oxidizing flame, any sulphides in the slag are completely burned out.

The rotary kiln commonly used ten years ago was 60 ft. long and 6 ft. in diameter. This has gradually been increased in length and diameter until the modern kiln is 140 ft. to 150 ft. long and 8 ft. to 10 ft. in diameter, and there are a few even larger kilns in use. Kilns are usually set at an incline of ¾ in. to the foot. With the lining and contents the modern kiln weighs 150 tons, and in revolving upon two bearings presents interesting constructional features.

In the case of the plant at Buffington, Ind., using 26,000 hp, situated between South Chicago and Gary, Ind., electric power is supplied from the steel works at these points. Each piece of machinery is driven by its individual motor. The high-tension line is connected through the cement plants, and the gas engines at these two steel works, 14 miles apart, operate continually in parallel.

This enables the cement plant to draw its power from either source or from both sources at the same time, as may be desirable at any time.

It has happened that one of these works has supplied power to operate the cement plant and has also furnished additional power at the same time to the steel works at the other end of the line.

The method of manufacture above described is the standard method of manufacturing Portland cement from natural deposits, and the finished product differs in no way from other Portland cements in chemical analysis, fineness, specific gravity, color, nor in the operation in practical work.

It has no peculiarities whatever and has no limitations as to its applications.

There is no difference, from the chemist's point of view, between the manufacture of Portland cement from natural deposits, such as limestone and clay or shale, and its manufacture from limestone and slag. Slag is really a mixture of the clay from the ore with the lime content of the stone used as a flux in the furnace.

Our method of manufacture of Universal Portland cement does not embody any real invention, nor is it based on any patents.

It is simply an adaptation to an artificial raw material of the regular Portland cement process formerly applied only to natural deposits.

True Portland cement in which slag is used as one of the raw materials should not be confused with Puzzolan or so-called "slag cements," which are simply mechanical mixtures of slag and slaked lime ground together without burning. Such cements are suitable only for use underground and in moist locations.

The manufacture of Puzzolan cements ni this country has practically been abandoned.

The remarkable growth of the Portland cement industry is not equalled by any other manufactured article. This is due to the economy, durability and plasticity of cement and concrete work.

While the large engineering work, such as dams, bridges, and heavy reinforced-concrete buildings consume large quantities of cement, the bulk of consumption at the present day is in a multitude of small uses. It takes an average shipment of only 5 bbl. a day to take care of the average customer of a large cement company.

For example, there is a steady increase in the application of cement to new uses on the farm, such as silos, fence posts, barn floors, feeding floors, watering troughs, corn cribs, etc. There, as elsewhere, concrete is rapidly displacing all forms of wood construction, this process being hastened by the continually advancing cost of lumber.

Beautiful effects are now being obtained in concrete surface finishes and its use in decorative work is advancing rapidly.

The use of Portland cement will continue to increase until the campaign of education of the small user has reached its finality.

Most of the problems of utilization of wastes or by-products of the blast furnace have been solved, and these solutions, in addition to being highly profitable, are powerful factors toward the conservation of our natural resources. Portland cement manufactured from slag, to a large extent, replaces wood; the waste gases displace coal, and reclamation of the flue dust conserves the deposits of iron ore.

The Problems of Tungsten Concentration.--II.

By H. C. Parmelee

Treatment of Old Tailings

The Cardinal mill of the Primos Mining and Milling Company is a remodeled gold mill, formerly known as the Boulder County mill. The tailings impounded during several years' run of tungsten ore before the canvas plant was installed are now being retreated at the rate of 20 tons per day. Naturally the grade of the material is not uniform owing to irregularities in its deposition and in the grade of the ores originally treated.

Ten stamps are used to break up the pulp and prepare a uniform feed for the concentrators. There is really little work for the stamps to do, and the operation is hard on them as there is little bed of ore to resist excessive wear of shoes and dies. The pulp is discharged through a 20-mesh screen and flows directly to one Wilfley table from which a fairly coarse concentrate is obtained. All of the tailings are returned by a Frenier pump to a Callow cone which thickens the pulp for tube mill feed. No screen analyses are available to show the work of the tube mill, but an inspection of the canvas plant would indicate that the sand is not ground fine enough, and that a heavier load of pebbles should be carried and the pulp further thickened to reduce the ratio of water to solids in the tube feed.

The tube mill discharge is pumped to a V-shaped settler, from which pulp is drawn by several syphons to a distributing box for three Frue vanners. All vanners thus receive the same feed and no attempt is made to classify the pulp. The vanner tailings are treated on the canvas plant which consists of 3600 sq. ft. of canvas arranged on stationary tables.

At the head of each set of tables is an inclined launder carrying the pulp to distributing boxes. One double box is provided for each pair of tables and wooden plugs are used to stop the flow of pulp to any box. Adjustable dividing strips are placed in the launder to equalize the flow of pulp to the several boxes. From the box the pulp is further distributed across the head of the table by means of several launders of varying length, and the flow over the table is still further equalized by strips of wood notched in a saw-tooth manner. A main service pipe for wash water extends along the upper end of the tables, with a valve and lateral pipe for each table. Two parallel launders, one for tailing and one for concentrate, extend along the foot of the tables and wooden aprons are so hinged that by shifting their position the pulp flowing off the table may be diverted into one launder or the other.

Another plant treating old tailings is that of Smith and Ardourell on Beaver Creek. The tailings from earlier work at the Wolf Tongue mill are hauled two miles and presumably treated at a profit. The average grade is reported to be about 2 per cent WO₃. Twenty tons a day are treated at this mill.

From the bins the pulp is conveyed by spiral conveyor to a shaking screen for the removal of stones and rubbish. It is then classified, the coarse grade being treated on a Wilfley table and the two finer grades on Monell slime concentrators. The Wilfley sand tailing is rejected and the middling reground in a Denver quartz mill. The Wilfley slime tailing and the Monell tailings are run over canvas. There are four canvas tables, each 4 ft. by 18 ft., built with a fall of 1 in. per foot in length.

Two grades of concentrate are recovered at this plant, instead of one as at the mill just described. The Monell slimer, which will be referred to in more detail later, will deliver two grades of mineral, one high and one low. The high grade from the Monell is combined with the Wilfley concentrate making a product running from 40-50 per cent WO₃. The Monell low-grade concentrate carries 30-40 per cent, and the canvas concentrate 10 per cent WO₃. If the Wilfley and Monell tables are running well the canvas plant makes a poor recovery and at times the concentrate is freely contaminated with pyrite and grains of gangue and scarcely worth saving.

When the present campaign on tailings is finished the mill will be fitted with crushing apparatus and adapted to treat crude ore. It appears probable that stamps will be installed, and that the capacity of the mill will be about 5 tons of ore per day as compared with 20 tons of tailings at present.

Treatment of Ore.

Before describing the other mills in detail it may be well to speak of the Monell slimer, as it is used in almost all of the plants and is regarded with favor. It is a slime concentrator of the traveling belt type with a shaking movement in the direction of belt travel. Its principal features are the unusual length and slow travel of the belt, and the large flat settling area afforded for the collection of fine mineral. The belt is 5 ft. wide and the working surface is 23 ft. long. The rate of belt travel is adjustable from 3 to 27 in. per minute, and is usually set at 18 in. The number of vibrations varies from 160 to 180 per minute. Pulp is fed at one side of the belt and tailing discharged at the other. The concentrate is collected in a box at one end in the usual manner.

The feed is first received in a classifying box set about 2 ft. above the deck of the table and near one end. Pipes from the several compartments of the box deliver the pulp to distributing launders arranged along the side of the table. The coarse grade is delivered near the end of the table where it forms a bed for the collection of the finer grades that are delivered farther on. Suitable wash water boxes are provided near the discharge end to wash off the sand and leave a clean concentrate.

The belt travels on strips of redwood laid lengthwise of the table. The upper or feed edge is slightly raised to direct the flow of pulp across the belt, and the lower or discharge side is inclined downward to aid in discharging the tailing. These inclinations at the edges of the belt extend for a few inches only, and the large area between them is perfectly flat, thus affording an excellent settling surface.

Owing to the manner of distributing the feed on the belt a bed of high grade mineral forms on its upper half, and a less well defined streak of lower grade mineral on the lower half. When the belt passes through the collecting box the mineral accumulates in corresponding beds of high and low grade material and the attendant keeps the two classes separate when cleaning the box. The illustration, Fig. 1, shows a Monell slime machine in the mill of the Tungsten M. & M. Co. on Beaver Creek. The bedding of high and low grade mineral is plainly shown on the end of the belt passing down to the concentrate box. The illustration also shows a Monell concentrating table which is of unusual design. The concentrates are drawn off through slots in the deck of the table and conducted by launders to boxes at the end.

The mill of the Tungsten M. & M. Co. has a capacity of about 5 tons in 24 hours. The ore is reduced by a jaw crusher to ½-in. size and fed to a Denver quartz mill fitted with 16-mesh diagonal-slot punched screen. The discharge is then passed through a 20-mesh screen, the oversize of which is treated on one Monell concentrator. The undersize is roughly classified, the coarse portions being sent to a second Monell table and the fine to a Monell slimer. There is no canvas plant at this mill and the tailings of all concentrators are run to waste. Two grades of mineral are recovered but no data can be given as to percentage recovery as there is no systematic chemical control of operations.

The Alton mill is a new plant modeled along modern lines, but as it has not been in steady operation nothing can be given but the plan of treatment. The ore is first crushed in a Samson No. 4 crusher, elevated to a bin from which it is delivered by a plunger feeder to a pair of 14-in. by 30-in. rolls. The roll product passes through a trommel fitted with 12-mesh screen, and the entire oversize is treated without further sizing on a double compartment Harz jig, having the same screen as the trommel. The undersize of the trommel and the hutch product of the jig are combined and treated directly on a Wilfley table.

Coarse concentrate is drawn from the side gates of the jig and the tailing is reground in a Denver quartz mill fitted with 20-mesh punched screen. The Wilfley tailings also are reground in the same machine. The reground pulp is pumped to an 8-ft. Callow cone, the underflow of which is divided between a Card table and a Monell slimer. The Card and Monell tailings are combined and treated on two canvas tables of 12-oz. duck, 6 ft. wide and 26 ft. long.
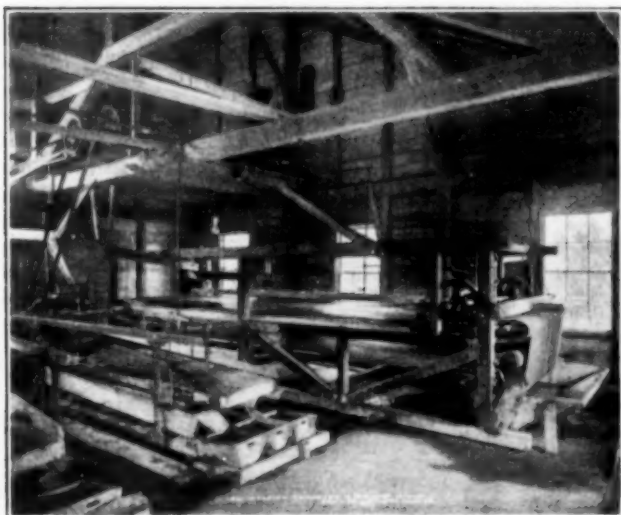

FIG. 1.—SHOWING MONELL SLIME CONCENTRATOR IN MILL OF TUNGSTEN M. & M. CO.

The manager of this plant expressed the opinion that the jig would make from 25 per cent to 40 per cent of the total recovery, according to the grade and physical condition of the ore. This belief is based on the treatment of 20 tons of ore when the mill was first put in commission.

The Wolf Tongue Mining Company whose mill at Nederland is shown in Fig. 2, is one of the two large companies that have been pioneers in the tungsten industry. The mill is an old remodeled stamp mill formerly used on gold and silver ores. As a result it is not as well arranged as a new plant would be, nor is its operation as satisfactory as that of a more modern plant. Nevertheless it shows very well the possibilities of coarse crushing and jigging tungsten ore, and it is probable that should the company erect a new plant it would conform closely to modern practice in ore dressing.

The company is at present treating custom ore which on an average will carry about 7-8 per cent WO₃. Each lot is sampled and bought on comparison of assays. The ore is first crushed in a Blake crusher and then in rolls set ½ in. apart. Power is applied to but one of the rolls which makes 90 r.p.m., the other running free. The roll product is elevated to a Vezin sampler which cuts out a fraction of the lot. This is further reduced by the tenth shovel method. and the portion thus obtained split on Jones riffles. The sample is then ground in a coffee mill, finished on the bucking board and divided into portions for the buyer and seller.

The reject from the Vezin sampler falls on a belt conveyor running to a trommel fitted with 16-mesh screen. The entire oversize is jigged, and the jig tailing is dewatered and sent to stamps for regrinding. The trommel undersize and the hutch product of the jig are combined with the reground tailing and classified for table treatment. The slime from the dewatered jig tailing flows directly to the canvas plant.

Ten 750-lb. stamps are used as regrinders of jig tailings. This battery formed part of an old equipment and probably is used only because it is convenient. The pulp is discharged through 12-mesh ton-cap screen and combined with the undersize of the 16-mesh trommel and the hutch product of the jigs. This mixed pulp is then classified in Flood classifiers and treated

on four Card tables. These tables make a fairly coarse high grade concentrate which is dried and sacked for shipment. The middlings and tailings are reground in another stamp battery through 50-mesh screen and concentrated on a fifth table.

The slime tailing from the fourth and fifth tables and the overflow from the fourth or last Flood classifier are collected in a distributing box and sent to three Monell slimers. A lower grade concentrate is recovered from these and their tailings are pumped to the canvas plant. On account of the preliminary coarse concentration, less reliance is placed on the canvas plant in this mill than in some others. It contains only 768 sq. ft. of canvas arranged in two tiers of tables, eight in a tier, each 4 ft. wide and 12 ft. long. A cement foundation was put in for the canvas and it appears to be perfectly satisfactory provided a firm foundation is made for the concrete work. The surface is smooth and there are none of the irregularities sometimes noticed in board surfaces where the boards have been laid either too close together and buckled, or too far apart and failed to close up on becoming thoroughly wet. Twelve ounce canvas is used and the tables have the usual fall of 1 in. per foot in length. Pulp is allowed to flow for three or four hours, after which the usual method of washing and cleaning is followed.

Cement settling basins for fine concentrates, with a cocoa matting filter at the outlet, are placed in the lower part of this mill. These basins have proved valuable in saving fine floating mineral from the canvas plant and from the concentrate boxes of the Monell slimers.

The Lakewood Mill.

In marked contrast to the principal methods employed in the plant just described are those practiced at the Lakewood mill of the Primos Mining & Milling Co. Those in charge of this mill and responsible for the methods employed are likewise pioneers in the milling of tungsten ore, and it is interesting to compare their methods with those of the other old company. The Lakewood mill is new and a model of construction as regards arrangement, convenience. cleanliness and facilities for saving time and labor. The illustrations, Figs. 3, 4 and 5, will tend to bear out these assertions.

At the head of the mill is ample bin room for the numerous

FIG. 2.—THE WOLF TONGUE MILL, NEDERLAND, COLO.

small lots of ore shipped by leasers when the company is in the market for custom ore. Separate crushing and sampling apparatus is installed for this work, consisting of jaw crusher. rolls and pipe sampler. The ore is bought outright on comparison of assays, and the lot is held until satisfactory settlement is made.

Ore from the company's mines is dumped into a large bin from which it is drawn over 2-in. grizzlies, the fine passing to the crushed ore bin. The coarse is reduced in a 10-in. by

16-in. Blake crusher and combined with the undersize of the grizzlies. The crushed ore bin is of 120-ton capacity, or about three times the daily capacity of the mill. Tappet-driven Challenge feeders draw the ore from this bin to the stamps through which all ore passes before concentration begins.

The battery frames are of heavy timber construction and the mortar foundation is built up of heavy timbers 18 ft. long set on end on a concrete foundation.

Concentration is divided into two principal departments; first, the customary table work with Wilfley tables and Frue vanners, and second, the recovery of fine mineral on the canvas plant. All of the concentrating tables are arranged on one floor below the stamps, and the canvas in three tiers below the tables. Concentrates from the Wilfley and Frue machines are combined. The product assays over 60 per cent WO₂.

The pulp from each battery of five stamps flows directly without classification to a Wilfley table. A clean concentrate is cut out and the balance is sent by a Frenier pump to a spitz-kasten which serves the first set of five vanners. The vanner tailings are combined and pumped to a large V-shaped box. The pulp is delivered to the box at one end through a pipe

FIG. 3.—THE LAKEWOOD MILL OF THE PRIMOS MINING AND MILLING
CO., LAKEWOOD, COL.

extending some distance beneath the surface, thus giving the coarse particles opportunity to settle. There are no partitions in this box, but as the pulp flows from one end to the other there is a rough classification due to the settling of the particles with the retardation of the stream.

Six spigot products are drawn from this box; the first or coarsest being thickened, reground in a tube mill and returned, and the other five treated on Frue vanners. The tube mill is shown in Fig. 5 in relation to the V-box and the Frue vanners. No screen analyses are available, but the statement is made that 85 per cent of the tube product will pass a 200-mesh screen. Pebble consumption is 50 lb. per day, but no figures can be given as to quantity of pulp ground. The vanner tailings and overflow from the V-box flow to the canvas plant.

The Lakewood mill has the most extensive canvas plant in the district, comprising 10,320 sq. ft. of cloth arranged in three tiers of tables. An idea of the area covered can be gathered from Fig. 3, showing the mill exterior. With the exception of the concentrating table floor, the entire lower portion of the mill is devoted to the canvas plant.

The construction of the distributing and collecting launders and other features of the plant is practically the same as described for the Cardinal mill of this company. The tables are cleaned once each shift and all canvas concentrates are collected in settling boxes on the lower floor. This is practically on a level with the drying house shown at the right of the mill in Fig. 3, where the concentrates are dried on steam tables and sacked for shipment.

The labor employed consists of six men on the day shift and three at night. During the day there is one man at the Blake crusher, two at the stamps and tables, two on the canvas and one in the drying and sacking room. At night there is but one man on the stamps and tables and two on the canvas.

The mill is steam heated and electric lighted. Electric power is received at 13,000 volts from the lines of the Central Colorado Power Co., and stepped down to 440 volts. Motors are
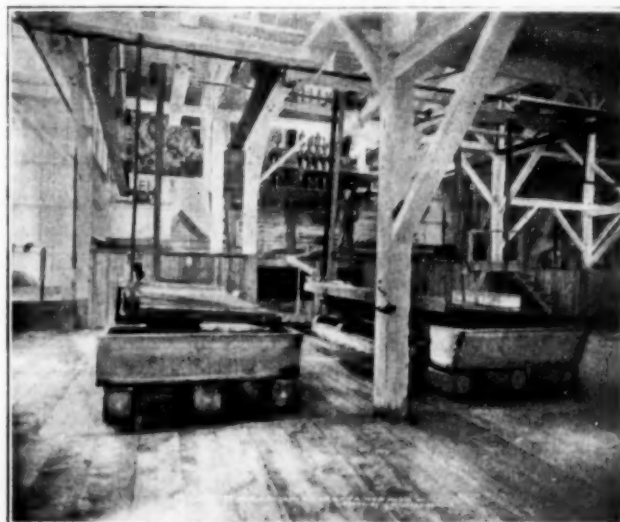
FIG. 4.—VIEW OF STAMP BATTERY AND WILFLEY TABLES IN
LAKEWOOD MILL.

distributed as follows: 15 hp, crusher; 35 hp, 10 stamps; 5 hp, 2 Wilfley tables and Frenier pump; 10 hp, 11 vanners and 2 Frenier pumps; 35 hp, tube mill.

Water is supplied from North Boulder Creek and delivered at the mill storage tanks either by a high line ditch, or by a Worthington 3-in. turbine pump direct connected to a 15-hp
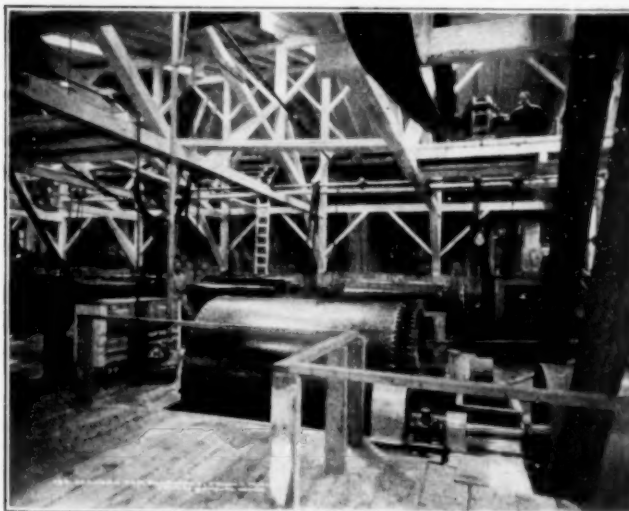
FIG. 5.—VIEW OF TUBE MILL WITH LARGE SETTLER AND VANNERS IN
BACKGROUND, LAKEWOOD MILL.

motor. The pipe line from the pump to the tank house is 700 ft. long and rises 85 ft. An unusually large quantity of water is used amounting to 250 gal. per minute for the treatment of 40 tons of ore in 24 hours.

All of the Boulder County tungsten mills are fortunate in having a supply of clear water which is of advantage in all ore dressing plants, but particularly where so much fine mineral has to be recovered by means of the canvas table.

The Development of Economical Ore Dressing Systems.*

By Frank E. Shepard.

In the quest for mineral values there appear to be several phases of development, one leading into the other. The discoveries of many of our large mining sections have resulted from the placer workings in which the values are found in gravels on or near the surface of the ground. The primitive methods with pick, shovel, and hand rocker often served to develop rich deposits. Then followed the more highly developed hydraulic mining systems involving greater expenditure of capital but bringing greatly increased returns. This trail of values leads to some source and so the patient prospector finally reaches an outcrop or vein which requires deeper workings with a more elaborate plant.

The search is first for high-grade ores, for the difficulties of mine development, transportation, and treatment are great and expenses in proportion. Bonanzas, however, are rare, and then must come the patient search for ways and means of developing better methods of mining, milling, and smelting for the recovery of values from lower-grade ores. Refractory ores are found which call for more advanced methods of smelting or the grade of ore is such that milling systems must be devised for concentrating into values sufficient for commercial delivery to smelting or refining points.

While the general trend of ore values has been from high grade to lower grade, there has also been a sombre accompaniment of decreasing market prices for the metals calling for serious study in all metallurgical systems. Ten years ago, manufacturers of machinery for ore reduction processes, in the Rocky Mountain section, were devoting much attention to the details in smelting machinery for the treatment of high-grade ores. To-day there is less call for smelting systems and the demand appears for improved ore dressing machinery for the recovery of values from lower and lower grade ores. This change from higher to lower grade ore conditions has taken place generally throughout the fields of gold, silver, lead, zinc and copper ores.

The stamp mill, with its amalgamating plates and bumping tables for the recovery of 50 per cent to 70 per cent of gold and silver values in the earlier period, is being succeeded by the concentration and cyanide plants producing recoveries of 90 per cent and over.

Lead-iron-zinc ores are presenting more and more difficult problems, and with the mining of lower grade ores of these metals comes the demand for better methods of crushing, screening, hydraulic classification and concentration, and in this particular line of ore treatment appears the great development in electrical processes of mineral separation.

The successful mining and milling of the great porphyry copper ore deposits of a grade as low as 2 per cent and under has brought about radical changes in milling methods and concentrating mills of 10,000 tons daily capacity have been successfully developed.

The tremendous pace in modern steel production has produced such demands on the iron ore supply that commercial possibilities have been found in the lower-grade iron ore deposits of Michigan and concentration plants have been installed for the treatment of 20,000 tons of ore daily.

The call for economy and reclamation of values in the Cripple Creek District has resulted in treating the dumps formed by the discarded ores of a former period by means of modern concentration and the cyanide systems at the remarkably low cost of $1.50 per ton.

Business men making investments in mining enterprises have found that it pays to be guided by specialists in this line of work. So many mistakes are made by the mining companies

rushing through the building of a mill before the mine is sufficiently developed. The professional fee of the competent geologist and mining engineer is well earned, and thorough examination and sampling of the mine should be completed to furnish sufficient data from which to design and develop a suitable mill system.

Having provided a representative sample of the ore to be treated, there are laboratories throughout the country where tests and experiments may be conducted and results obtained from which an economical system of ore treatment may be planned. The various manufacturers of mining and milling machinery offer engineering advice in the development of systems of ore treatment, and while this may be competent and the result of good practical experience, such advice depends upon data furnished by the mining company itself.

Large mining companies appreciate the value of preliminary investigations of ore conditions in a mine, and the manufacturer has competent advice from which to design and construct suitable machinery and equipment for the ore dressing system; but in the smaller mining organizations, the building of a mill oftentimes appears to be more important than the development of ore bodies and the manufacturer is called upon to furnish machinery to treat ores which may change radically in further development of the mine.

In extensive mining operations the plan is followed of erecting mills to test the practical application of the mill system determined upon, and this proves of great value in solving difficult problems of ore treatment.

In the smaller mining companies, often composed of business men without any experience in mining and milling methods, it is seldom found that sufficient funds have been devoted to these important preliminary tests and instead of employing competent technical advice some one of the directors of the company who "has a liking for machinery" assumes charge of affairs and the resulting mining and milling composition ends in minor chords. However small the proposed mining or milling installation, it is alway advisable to secure the advice of the mining or metallurgical engineer who studies the entire field of operation and the economic conditions relating to mine and mill, makes suitable examination of mine, takes sufficient samples of the ore, and after preliminary tests is prepared to submit general plans and specifications for a suitable mill system.

While the manufacturers of mining and milling machinery are by custom expected to prepare plans for milling systems, on account of the stress in business competition, such is not an economic condition, and it is better practice to have one engineer thoroughly advised on all technical matters from inception to the complete milling plant.

The manufacturer has the very important part to perform of attending to the many details of mill installation and it is his affair to keep up to the minute in all the improvements which aid in reducing costs and increasing recovery of values from the ores.

Mill design is a profession in itself. It is not necessary for the consulting, mining or metallurgical engineer of the mining company to be an expert in the details of mill construction as the machinery manufacturers make a study of these problems. The detail drawings of the mill should be intrusted to an engineer who has not only a very thorough knowledge of mill machinery details but an intimate knowledge of mill men and practice.

In the selection of mill machinery the new inexperienced mining company usually buys the cheapest grade, while the mill superintendent of long experience buys the best machinery that can be built. In a small mill treating 50 tons in 24 hours of an ore having a value of $15 per ton and assuming the mill recovers 80 per cent. of the value, a delay in operation of but one hour means a loss of $25. It is manifestly poor economy to save a few hundred dollars in the original investment at the expense of several thousand dollars in delays and repairs.

Ore crushers were formerly equipped with hard cast-iron jaw

plates which were required to be removed every few weeks, whereas now these crushers are provided with the best crucible-steel jaw plates which last for several months. Crushing rolls were formerly equipped with cast-iron shells which gave short service and soon became grooved; now the best forged-steel, machine-finished shells are used with the result of longer service and better product.

Elevating and conveying apparatus has reached a high state of development, all tending to greatly reduce costs of handling materials. By means of these excellent modern devices the entire mill system is automatic from beginning to end, save for the adjustments of machinery.

One of the greatest if not the greatest improvement in the wet concentration of ores has been the important development in the preparation of ore pulps previous to jigging and table work as a result of the investigation of Dr. Robert H. Richards of the Massachusetts Institute of Technology.

Dr. Richards' devices for hydraulic classification of ore pulps, previous to table concentration, have produced savings in ore values over former methods amounting to hundreds of thousands of dollars annually.

If we refer to our small mill before mentioned having 50 tons capacity daily and recovering 80 per cent of values in $15 ore, an improvement in recovery of only 5 per cent will mean an additional annual saving of over $12,000. Such improvements have actually been accomplished in practical mill systems using Dr. Richards' systems of hydraulic classification.

The tube mill is an important element in modern mill systems and was adapted from cement mill practice. It takes the place of the more complicated crushing machines involving the frequent renewal of shoes, dies, screens, and various parts in roller or chilian mills and presents a simpler and more economical means for the fine crushing of ores. The development of the tube mill in connection with the stamp mill in South African practice is extraordinary; the combination effecting an increase in tons crushed per stamp from five tons, before the tube mill was introduced, to 20 tons per stamp when assisted by the tube mill.

The better understanding of the preparation of pulps previous to table treatment as well as the better understanding of the concentrating tables themselves has brought about great improvement in mill recoveries.

The development of electromagnetic and electrostatic concentrating machinery has established practical separation of iron and zinc minerals which before was most difficult or impossible.

Most interesting use has been made of the property of film tension in liquids for the purpose of separating mineral values from the attending gangue. In general the sulphides of the metals may be floated on the liquid while the gangue sinks. While the system is limited to certain conditions, it opens a fascinating field and one which may develop into great possibilities.

Even greater progress could be accomplished were it possible to obtain the earnest co-operation of mill operators. In some cases splendid triumphs have been accomplished because new devices have been given the "square deal" by intelligent and progressive mill operators. On the other hand, many a mill system is undeveloped on account of personal prejudice and lack of initiative.

Failures in mill systems are often due to neglect in making tests to determine the points where losses occur or the costs of various operations. Changes suddenly occur in the ore and if not promptly followed up with required changes in the mill system will result in serious losses.

One of the modern requirements is thorough sampling and the best mills make use of automatic sampling throughout the system.

It is most important to have the right spirit prevail among the members of the mill crew. Good team work tells here as well as in the game, and a fair attitude and attention to im-

proved methods of milling means also thousands of dollars to the mining company.

We often see machinery advertised as "fool-proof." If the mill system is to be run by fools then we need the fool-proof machinery, but if we cannot advance beyond the stage of fool-proof machinery then we might as well give up our cherished ideas of additional recoveries in ore values. For the economical recovery of values in low-grade ores we must expect greater complication in mill systems, but if we can save $25,000 by the addition of some machine, even though not classified as "fool-proof," is not the expenditure warranted of $2,500 in wages to some millman who will be a friend to the machine?

The man who stands by the machine has the opportunity of discovering many points, possibly improvements, regarding the operation or results of that machine. If the personal equation of that man is plus and he receives proper recognition by the management, the result is a substantial increment to favorable mill devolpment. Mills are usually located in remote sites, difficult of access and in many cases at altitudes of 10,000 and 12,000 feet above seat level. It requires some financial incentive to induce good men to work constantly under these conditions and liberal wages must be offered to establish a good order of intelligence in the mill crew.

The study of mill systems has in some cases shown a high order of inventive ability among mill superintendents in the development of devices for increasing the recovery of ore values, or reducing the expense of operation. There is, however, a great amount of blind prejudice to overcome when an attempt is made to introduce improved methods in the milling systems. This is a strange condition, for one would think that any millman would be favorable to any device which would show improved results even at the expense of some extra attention, but the fact remains that it requires a bitter fight to establish new systems.

A very interesting psychological fact exists in mill conditions. If a new device or system be introduced by some one of the mill attendants and he has the care of the device, that device is certain to work successfully. Should a system be introduced by the manufacturer, even though such a system may have been proven a success, all manner of prejudice and resistance appears and a well-merited system is classed as a failure.

If our graduates of technical schools could be induced to serve a few years of apprenticeship in the splendid experience of these mill systems and devote their technical training toward assistance in the development of improved devices, instead of obstructionary measures, there would be good progress made for better ore dressing systems.

With the successful treatment of low-grade ores there must be better study of conditions, better skill in operation and a constant effort for improvement. The modern mill calls for a higher order of materials of construction; concrete and steel are now being used when a few years ago such use was designated as extravagance. Better machinery is being used and the use of higher grade materials throughout the mill construction is considered a necessity.

The opportunity for improved results in ore dressing systems is great. When we know that in the operation of 10,000-ton copper ore concentrators, the loss on every ton of ore treated is 30 per cent to 40 per cent; that in precious metal wet concentrating mills the loss is from 20 per cent to 30 per cent, the time is now at hand when the data from long lines of investigation assisting the engineer and the builder, added to the practical application of the honest millman, should produce some telling results.

The trail of the gold seeker leads along difficult paths and through discouraging obstacles. Constant search and persistent effort will be the procedure as long as there remains a quest for precious metals—but the problems call for work and work develops the man.

"Let not him that seeketh cease from his search until he find, and when he finds he shall wonder; wondering, he shall reach the Kingdom, and when he reaches the Kingdom he shall have rest."

Denver Engineering Works, Denver, Colo.

Notes on the Analysis of Tungsten Ores.

By Hugh F. Watts.

Insomuch as tungsten can be readily separated from other elements by precipitation as yellow oxide or as mercurous tungstate, and as these precipitates can be easily converted into weighable tungstic oxide, few methods depending on other reactions have been proposed for its gravimetric determination in ores.

Volumetric methods have not come into general use principally for the reason that most of the time consumed in making the assay is in getting the ore decomposed and the tungsten separated from iron and silica. Solution and titration of the tungstic acid would not, therefore, present any advantages over weighing.

An interesting method described by Ekeley and Kendall (*West. Chem. & Met.*, Jan., 1908) consists in dissolving the tungstic acid from the silica by means of a measured volume of standard ethylamine and titrating back the excess with standard oxalic acid using phenolphthalein as indicator. The method has given excellent results.

Other volumetric methods depend on the principle that tungstic acid can be dissolved in standard sodium hydrate and the excess titrated with standard acid.

For the purpose of analysis, tungsten ores can be decomposed by fusion with alkaline carbonates; by boiling in solution of sodium hydroxide; by digestion in a mixture of hydrochloric and hydrofluoric acids, or by digesting in aqua regia.

When decomposition has been effected by fusion or digestion with alkalis, it is essential that the tungsten be precipitated as mercurous tungstate. A method given in many text-books depending on the separation of the yellow oxide from solutions of alkaline tungstates by evaporation with hydrochloric acid is unreliable, as it seems to be quite impossible to recover all the tungsten in this manner even when the evaporation and boiling with acid is repeated many times.

When the separation is made by mercurous nitrate, it should, of course, be remembered that phosphorus, vanadium, molybdenum, etc., are also precipitated. Moreover, even though these interfering elements are absent, as is generally the case, the voluminous precipitate of mercurous tungstate is apt to include impurities which cannot be easily found and accounted for, so that the results are apt to be a little high. Nevertheless, this is an excellent and much used method and is probably the only one that can be used on some ores.

Hutchin and Tonks (Inst. of Min. & Met., London, May, 1909) have described a method of especial use for battery pulp and tailings, the decomposition being effected by hot caustic alkali solution. The charge taken may be from 5 grams upward, which is digested in a porcelain dish with 25 cc of a 25 per cent solution of caustic soda. From this point the assay is finished as though the decomposition had been made by fusion. The method is not applicable to ores containing scheelite.

A mixture of equal parts of hydrochloric and hydrofluoric acids will decompose tungsten ores, sometimes to an almost complete solution. If now an excess of hydrochloric acid is added and the hydrofluoric acid expelled by evaporation, the yellow oxide will be precipitated on the addition of water. (Method of O. P. Fritchle.)

Conflicting statements can be found regarding the solubility of tungstic acid in hydrochloric acid. The unignited precipitate is certainly soluble to considerable extent in strong hydrochloric acid. Upon dilution with several volumes of water the tungsten is completely precipitated on standing. If subjected to filtration with the proper precautions in washing and the filtrate is perfectly clear and bright, no tungsten can be found therein or at the most in but very slight amounts.

Tungstic acid cannot be washed with pure water as it forms a pseudo-solution and causes the filtrate to be turbid. The wash water must contain acid or a solution of some salt. Similarly, in separating tungstic acid from silica with ammonia, much silica is carried through in a turbid stream when ammonia water is used alone. The presence of a little ammonium chloride prevents this in a great measure, but not to the extent, however, of making a complete separation, as silica is appreciably soluble in ammonia. The use of ammonia is, nevertheless, a most convenient means of separating all the tungstic acid from the bulk of the silicious residue.

When it is expected to use ammonia in the separation, care must be taken not to carry the assay to hard dryness as the tungstic acid is thus rendered difficultly soluble. Even prolonged heating on the water bath makes its solution uncertain.

Tungstic acid and silica may be separated by evaporation with hydrofluoric acid, the addition of sulphuric acid being unnecessary.

A Bunsen or Teclu burner can be safely used to ignite tungstic acid for weighing. The use of a blast lamp is inadmissible since tungstic acid is slowly but steadily volatilized at such temperatures. (Wells and Metzger, *J. Am. Chem. Soc.*, Vol. 23, No. 5.)

When working with material so intractable as tungsten ores, whatever method of analysis is adopted, it is absolutely essential that the sample be finely agate-ground.

Having regard for some of the foregoing principles, the methods in practice are carried on about as follows:

Fusion Method.

From ½ to 1 gram of the sample is thoroughly fused in a platinum crucible with 4 or 5 grams of sodium carbonate. The melt is dissolved in hot water, a little sodium peroxide added, and the iron and manganese separated by filtration. The tungsten passes into the filtrate as sodium tungstate.

Acidify the filtrate slightly with nitric acid, make slightly alkaline with ammonia, boil and filter from the usually slight precipitate. To the filtrate add a few drops of methyl orange and titrate with nitric acid, contained in a burette, until just acid. Heat and add an excess of a clear solution of mercurous nitrate. Now add a few drops of a strong solution of sodium carbonate to precipitate a little mercurous carbonate. Remove from the heat and allow to settle for a few minutes. Filter on a 12-cm filter paper and wash well with hot water.

Without drying the precipitate further than can be done on the filter pump, transfer it with the filter to a platinum crucible. Char the paper at a low heat and finally ignite, in a good hood, over the flame of a Teclu burner. Cool the crucible, moisten the contents well with hydrofluoric acid, dry, reignite and weigh as tungsten oxide.

Aqua Regia Method.

Usually 1 gram of ore is used for analysis. If extremely high grade or a concentrate, ½ gram is sufficient, while for mill tailings, etc., 2 grams or more may be taken. For high-grade ores 50 cc aqua regia are used, for low grade 30 cc. The decompositions are made in 4-oz. covered beakers. The degree of heat should be such that there is a gentle action of the acid on the mineral, but the solution should not actually boil. The assay should be occasionally well stirred up to prevent caking. When the solution has evaporated to 10 cc or 15 cc it is removed from the hot plate and diluted with 50 cc hot water and allowed to stand for half an hour. Decant the clear solution through a good filter paper. The residue settles readily and can be decanted close. Wash the residue twice more by decantation, using each time 50 cc hot water, which has been acidulated with hydrochloric acid, keeping as much of the residue in the beaker as possible.

Add about 20 cc ammonia solution to the beaker to dissolve

the tungstic acid and decant the solution through the filter. The ammonia wash solution is made by mixing 200 cc strong ammonia with 1000 cc distilled water and adding 10 cc hydrochloric acid to form a little ammonium chloride. The residue of silica is now brought on the filter and washed thoroughly with the ammonia solution, which is most conveniently used in a wash bottle.

The resulting silica should be white; if it shows any particles of undecomposed mineral, as may sometimes occur in the high-grade ores, from the protecting action of the separated tungstic acid, instead of bringing on the filter, wash two or three times by decantation and treat again with aqua regia, using 15 cc or 20 cc, which will generally complete the work in a short time. One trial is all that is usually necessary with the ordinary run of ores.

The ammoniacal solution is now evaporated to dryness in a weighed platinum dish, ignited gently at first to drive off ammonium salts and finally at the full power of the burner. Cool, moisten with hydrofluoric acid and again evaporate to dryness and ignite. Weigh as tungstic acid.

The aqua regia method is the one used in the writer's laboratory for the valuation of ores from the Boulder district and has been found to work well on any of our western ores. Two time-saving devices are used to facilitate the work, a McKenna mechanical agate grinder, which is a most efficient machine, and a special air bath for evaporating the ammoniacal solutions, constructed so that each dish has its own receptacle. Each receptacle is heated from beneath with a small Bunsen burner and overhead is furnished with a glass tube through which a current of air is made to impinge on the surface of the liquid. When removing a dish the tube may be turned to one side out of the way.

The time that this arrangement saves by hastening the evaporation is very material, and insomuch as the evaporation can be started while the washing is still in progress, this part of the operation is really very short. The air is furnished from the blast arrangement of a Richard's filter pump.

Tungsten ores are paid for by the unit of tungstic acid, consequently the weight derived from the assay gives at once the figure without calculation further than reduction to percentage. If metallic tungsten is wanted the factor is 0.7931.

A simple qualitative test can be made by boiling a little of the finely ground mineral with 10 cc strong sulphuric acid in a test tube. When cold the liquid assumes a deep blue color. The addition of tin or other reducing agent is needless.

The papers cited in these notes will be found of value to any one interested in tungsten analysis.

Boulder, Colo.

Progress in Mexican Metallurgy During the Century of Independence.*

By A. GROTHE.

On reaching the patio we find a scene vastly different from that of the morning. A large cavalcade has arrived and the dismounted riders are busy examining the numbered ore heaps. It is the day appointed for the ore sale or "rescate" and the visitors are the owners or managers of the many reduction works in the valley near town, all intent upon purchasing ore for their "haciendas." Notebook in hand they proceed from heap to heap, judging the weight by the eye and the contents by panning in a small lacquered gourd (jicara). They are won-

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*In the days of rapid progress in the twentieth century it is difficult to recall or imagine some of the conditions which existed a century ago, or the crude methods then employed for the valuation and treatment of ore. At a recent meeting of the *Concurso Científico y Artístico del Centenario*, held in Mexico on Feb. 25, 1911, Mr. ALBERT GROTHE represented the Mexican Institute of Mining and Metallurgy, and read a paper showing the progress of metallurgy in Mexico during the past century. In a delightfully interesting manner he contrasts the conditions existing at the Mellado and Rayas mine at Guanajuato in the year 1809 with those of the present day. His full address, of which the following is an abstract, appears in the *Bulletin* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 2.

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derfully expert at the operation (tentadura) when dealing with ore of which they have experience.

The art of assaying is hardly practised as yet (any more than that of mine surveying) and in consequence grievous mistakes occur occasionally; as, for instance, when some years later ore was found in abandoned workings in the upper levels of Valenciana which had the look of compact iron pyrites and gave no indication of silver in the tentadura. Nevertheless it was found by fire assay to contain from 22 to 24 marcos of silver per "montón" (3.5 to 4 kg per ton). Still these mistakes are the exception, and ordinarily the estimate of weight and contents is astonishingly correct.

When all are ready with their calculations the manager of the mine (administrador) takes his position near one of the heaps and invites bids for this number. The competing buyers approach him one by one and make their bids in whispers. The proprietor of the mine or his representative do the same as a protection against possible collusion among the rescatadores. When after the auctioneer's solemn: "No hay quien diga más?" no further offer is forthcoming, the name of the highest bidder and the price to be paid are announced, and the group moves on to the next montón until all are sold. In this manner thousands of dollars worth of ore are sold every week. Losses are made, and great bargains secured, and it seems that this gambling nature of the method makes it so attractive to those interested that it has survived in Guanajuato to until the end of the last century, when the San Cayetano Company still sold the production of its mines in this manner. And, crude as the system was, perhaps no great injustice resulted to either buyer or seller.

The writer had the ore offered for sale by those mines in 1888 carefully weighed and assayed and found that at the end of the year the computation of the value on this correct basis differed only 2 per cent from the price realized, the latter being the higher. He thereupon published his figures on the day before the rescate and sold the ore on the basis thereof. This put an end to the picturesque but risky proceeding of former times.

At the conclusion of the rescate the mules which have to convey the ore to the reduction works swarm in, and the arrieros proceed to load the ore in leather sacks, while their masters sit down to a sumptuous lunch, after which the tables are cleared for a friendly game of cards, which is often prolonged far into the night, in the course of which many of the purchased heaps of ore change hands.

Not being possessed of the gambling spirit, we shall rather follow the ore to the reduction works.

At the hacienda it is weighed, without, however, a determination of the moisture being made. This and other inaccuracies make the returns as to results obtained not quite so trustworthy as might be desirable. The coarse ore (grueso) is then crushed to pea size in crude wooden stamps or equally primitive roller mills with stone runners, and, after being mixed with the smalls, is fed to arrastres, where it is ground to impalpable slime under addition of water.

The arrastre is a circular stone tub of about 9 ft. diameter half sunk below the floor and provided with a bottom of hard basalt or porphyry, the stones being placed on end. The grinding stones are large pieces of the same material suspended by chains from cross-bars fixed to a vertical shaft, to which a mule imparts a rotary motion. Eighty or more arrastres are placed in a row in a large shed or galera. The bottom of the arrastre is covered with a thin layer of silver or copper amalgam to which the fine particles of metallic silver and gold contained in the ore adhere.

Every morning the arrastres are emptied of the ground pulp, which goes to the "patio" for further treatment. The capacity of an arrastre is from 7 to 8 cargas (1100 to 1300 kg) per day, according to the degree of softness of the ore. Every two to three months a "raspa" is made; that is, the amalgam is removed from the bottom and joints. The scrapings are thinned

out with mercury, filtered and the remaining mercury distilled off in vertical iron retorts or "capellinas." The resulting bullion, which is very rich in gold, is then melted in charcoal furnaces, cast into bars and taken to the mint, where it is assayed and parted—that is, the two metals are separated, and obtained in an almost pure state. By this process about one-half of the gold contained in the ore is recovered, the other half remaining in the pulp is lost, as we shall see.

The pulp is collected in flat heaps, called "tortas," each containing about 100 tons of ore, on the flagged floor of an open yard, called the "patio," from which the name of the whole process is derived. It was invented by Bartolomé de Medina and first practised by him in Pachuca in 1557. It was admirably adapted to the available resources and the requirements of the country, and, crude as it may appear to the modern metallurgist, has been of incalculable benefit to the mining industry.

The proper proportion of common salt is then added to the torta and thoroughly mixed with the pulp (repaso) by the action of twenty-four horses held on lines by the "tortero," round whom they circle, while he himself continually changes his position so that all parts of the torta are eventually reached. Before 1793 this work was done by hand.

The next operation consists in the admixture of "magistral," an impure sulphate of copper obtained by the calcination of copper pyrites containing variable quantities of iron. Lastly the torta is incorporated—that is, mercury is spread over its surface in a fine spray produced by filtration through porous cloth. After each addition of ingredients, and afterwards every other day, a repaso is given.

In from twenty to thirty days, according to the size of the torta and the weather, the chemical action is complete and the torta is said to be "rendida." The silver and gold have combined with the mercury to fluid amalgam, from which the pulp is washed by a liberal addition of water and stirring in large cylindrical vats. It is now only necessary to filter the excess of mercury off the solid amalgam, distill the latter in the manner described before the gold amalgam, and melt.

The product is very pure. Its fineness is determined in the mint and expressed in thousands of the gross weight. Impurities are deducted and it should be noted that if gold is present in less proportion than three-one-thousandths it is not only not paid for, but even deducted from the silver as an impurity. Small wonder that Mexican pesos of this period from the gold-bearing districts are subjected, at a good profit, to parting in London and Paris.

The patio process is a purely empirical one and requires great practice on the part of those who carry it out. It would lead us too far were we to attempt a detailed description of all the operations in connection with it, or to mention the various vicissitudes to which a torta may become subject, and the means for diagnosing and correcting them. The only guide for the "azoguero" is that afforded by frequent tentaduras.[1]

From time to time improvements, based upon chance discoveries, have been introduced and local modifications demanded by peculiarities of the ore have been made, but the history of the process is free from the often absurd nostrums frequently used during the earlier times of Californian panamalgamation. Altogether Medina's discovery was an admirable one and much more perfect than the mining methods in use at the beginning of the nineteenth century, which have been described above, for no other process until the advent of cyaniding has been able to oust it.

We have tried to produce an impressionist picture in broad traits of the methods employed in Mexican mining and metallurgy at the beginning of the century, in the years immediately preceding the outbreak of the revolution which ultimately liberated the country from the oppression under which it had been ground for more than two centuries and a half. It was the most productive period of Mexican mining, with which the general prosperity of the whole country had naturally kept pace.

Another discovery which gave an immense impulse to Mexican mining was the method of extraction of the precious metals from their ores by means of weak solutions of potassium cyanide. There has been much controversy over the question as to who was the real discoverer of the principle underlying this operation. We shall not enter into it here. Uncontrovertible, however, is the fact that since the introduction of the process by the McArthur-Forrest interests millions of tons of ore which formerly had to be considered as of no value can be treated profitably, in consequence of which the economic conditions of the mining industry have been radically changed and much foreign capital has been attracted.

Mexico was somewhat slow in adopting this method, possibly on account of the fact that as at first practised it was more applicable to gold than to silver ores.[2]

In our description of the process we can be much more concise than we were in that of the earlier methods, if for no other reason than that Rayas in 1808 is immeasurably farther from us than, for instance, Pachuca in 1911. Any of the modern reduction works there can be readily visited by those desiring some knowledge of this newest branch of metallurgy of the precious metals. Such a visit will afford convincing evidence of the great importance of this method of ore treatment and will show that, while Mexico entered the field long after other countries, once the start was made it has not only kept pace with them, but, certainly as regards the treatment of silver ores, is ahead of most.

Briefly, the process is carried out as follows: As in any other wet method of extraction, the ore is finely triturated in stamps, Chilean mills and tube mills, cyanide solutions being added from the beginning. The separation of sands from slimes, which a few years ago was considered indispensable to successful cyanide practice, has been superseded in all modern plants of this country by the "all-sliming" method, in which the sand is all ground as fine as possible, or, rather, as fine as necessary to obtain the best economic results.

The pulp, consisting of finely ground ore and cyanide solution, is then agitated by means of mechanical stirrers or by compressed air, the latter now being the rule in Pachuca. When the cyanide has completed its chemical action on the gold and silver of the ore, these substances are found in solution as double salts with sodium cyanides. This solution must now be separated from the worthless tailings by means of vacuum or of pressure filters of various construction, of which new forms are almost daily brought out. The clear, filtered solutions are then brought into contact with zinc shavings or zinc dust, on which the precious metals are precipitated at the expense of the zinc.

The solutions, after having been brought to their original strength by the addition of fresh sodium cyanide, enter again into circulation by being returned to the grinding apparatus. The black precipitate obtained by the operation, consisting chiefly of gold, silver, finely divided zinc, lime, alumina and other impurities, is then washed in filter presses, dried, melted with suitable fluxes, and finally cast into bars of about 30 kg, which are exported. The product is not of the same purity as that from the patio process, rarely being above 950 fine, but, on the other hand, the extraction is above 90 per cent of the silver and 98 per cent of the gold, and the process requires only as many days as formerly weeks, and the cost is less than one-fourth.

A description of the mechanical details of the appliances used in the cyanide process would fill volumes and not be pertinent

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[1] The old system of tentaduras as a guide in the patio process was gradually replaced by assays of the amalgam, proposed in 1875 or 1876 by Manuel María Contreras, M. E. The art of making correct tentaduras has well-nigh been lost among the younger generation of engineers.

[2] In 1894 the McArthur-Forrest Company installed a movable plant for the treatment of tailings in the State of Sonora which they shifted from one point to another. This was followed by stationary plants in Guanajuato, El Oro, Pachuca and other mining centers. In Pachuca the new plants followed each other rapidly. The last mill which has been reformed is that of Purísima Grande, where centuries ago the system of Bartolomé de Medina had originated and where it was practised until very recently.

to the purpose of this paper, which is to draw attention to the great and beneficent change in methods since the time when Mexico obtained its independence, especially during the last thirty years, when continued peace made it possible to devote all the available energy to the development of the industry.

It must be noted that since the introduction of cyaniding all efforts have been directed toward improvements on mechanical lines, in which certainly wonderful progress has been made. Our knowledge of its chemistry has, however, not advanced at the same rate and is still very much as it was when McArthur-Forrest first advocated and practised the principle involved, with the important exception, however, that it was found that in the treatment of silver ores the extremely weak solutions used on gold have hardly any effect, but have to be considerably strengthened, a fact which was not known at that time. Before this was known successful treatment of silver ores by cyanide was considered impracticable.

Various other chemicals in combination with cyanide have been proposed and tried, as they were supposed to accelerate or intensify the action, but the largest number of them have no effect or are only of use with certain classes of ore. Mention should, however, be made of Clancy's proposal to add to the straight cyanide solution calcium cyanamide, an alkaline sulphocyanide and an alkaline iodide, and passing a low-voltage current through the agitated pulp. The process has not as yet been fully tried on our Mexican ores, refractory on account of containing part of the silver in combination with manganese.

If successful on these, many ores not amenable to straight cyanide treatment could be profitably worked in this manner. According to the experiments of the inventor even in ordinary ores a saving of cyanide is obtained as the solutions are regenerated at the expense of a less costly substance and a trifling expenditure of electrical power. If the claims made for this process are substantiated by results on a working scale, new prospects for the industry will doubtless be opened.

We approach the end of our task, which was to indicate in broad lines the difference between the methods in use at the beginning of the century of independence and those now practised. Some of the manifest faults of our sketch are due to the disproportion between the space at our disposal and the great mass of material which imperatively seemed to demand consideration. This caused a constant conflict between the desire for completeness and the necessity for brevity, which perforce made the work fragmentary. We have dwelt more minutely on the early part of the period under consideration than on more recent occurrences. The knowledge of these early times is rapidly being obliterated; in none of the splendid and valuable treatises on old Mexican mining history is it presented in a connected form, being generally overlaid by a wealth of statistics and detail, which to the non-technical reader is not calculated to increase clearness.

We are safe in concluding with the statement that, especially during the period of peace and security of the last thirty years, the progress has been enormous and in expressing the hope and conviction that under a continuance of these conditions the industry will gain in importance at the rate justified by the large mineral wealth of the country.

Asbestos of Commerce.

The asbestos of commerce includes fibrous minerals of several species, but all may be included under amphibole and serpentine. Fibrous amphibole is properly called asbestos, fibrous serpentine is chrysotile, and the especially fine silky form of both amphibole and serpentine is known as amianthus, but in the trade all are included under the general term asbestos. Like auriferous quartz and the ores of other precious metals, asbestos generally forms veins, but in some places where it is of low grade it makes up the whole mass of the country rock. The three types of asbestos, cross, slip and mass fiber, are distinguished by the form of aggregation.

Treatment of the Anode Sludge in Electrolytic Refineries.

From lecture notes prepared for the School of Mines, Columbia University, by EDWARD F. KERN, PH.D.

Anode sludge is the residue of the anode, formed during the electrolytic refining of metals. It consists mainly of the associated impurities which stand lower in the E.M.F. series than the metal which is being refined.

Analyses of copper anode sludge vary between:

15.0 to 55.0% Cu (as Cu: Cu_2S: $CuSO_4$: Cu_2Se: Cu_2Te: etc.)
2.0 to 8% Sb (as Sb_2O_3:$(SbO)_2SO_4$:$Cu_2O.Sb_2O_5$:etc.)
1.5 to 6.0% As (as $As_2O_3.H_2O$: $Cu_2O.As_2O_5$: etc.)
0.2 to 8.0% Bi (as Bi_2O_3: $(BiO)_2SO_4$).
0.5 to 6.0% Pb (as $PbSO_4$).
0.5 to 12.0% S (as Cu_2S).
0.5 to 1.5% Fe (as $FeO.FeSO_4$).
0.1 to 2.5% Se (as Ag_2Se: Cu_2Se).
0.1 to 3.5% Te (as Ag_2Te: Cu_2Te).
5.0 to 50.0% Ag (as Ag : Ag_2Se : Ag_2Te).—1,500 to 15,000 oz/T.)
0.017 to 0.70% Au (as Au) = (5 to 10 oz/T).

Typical analyses of sludge from Montana copper:
40.% Ag (12,000 oz/T) : 2.% Au (600 oz/T) : 25% Cu: 5.% Se & Te: 10% As + Sb: 18% PbSO4, S etc.

Analyses of lead anode sludge vary between:

6. to 18% Pb (as Pb, PbSe, PbTe, oxide).
5. to 25% Cu (as Cu, Cu_2Se, Cu_2Te, oxide). ⎫ The OXIDES
5. to 70% Sb (as Sb or Sb_2O_3). ⎪ formed during
2. to 50% As (as As or As_2O_3). ⎬ drying of
0. to 20% Bi (as Bi or Bi_2O_3). ⎪ the SLUDGE.
0. to 2% Sn (as Sn or SnO_2). ⎭
0. to 3% Se (as Cu_2Se, Ag_2Se, etc.).
0. to 4% Te (as Cu_2Te, Ag_2Te, etc.).
10. to 40.% Ag (as Ag, Ag_2Se, Ag_2Te, etc.) = (300 to 12,000 oz/T).
0. to 1% Au (as Au) = (0 to 300 oz/T).

Typical analyses of sludge from trail, B.C., lead:
32% Ag (9,600 oz/T) : 0.3% Au (90 oz/T) : 9.0% Cu: 27% Sb : 7.% As : 14.% Pb : 11.% moisture, O., etc., and 19.% Ag (5,700 oz/T) : 0.1% Au (30 oz/T) : 11.% Cu: 31.% Sb : 9.% As : 16.% Pb : 14.% moisture, O. etc.

A characteristic of lead anode sludge is that the constituents are present principally as metals or oxides, whereas the constituents of copper anode sludge are present principally as basic sulphates, sulphates and oxides.

A reason that basic sulphates are the predominant constituents of copper anode sludge is that the principal impurities (Pb, Sb and Bi) of crude copper stand above copper in the E.M.F. series, so are dissolved along with copper, but forming stable sulphates and basic sulphates they are precipitated and collect in the sludge as such.

The impurities of base bullion (crude lead) stand below lead in the E.M.F. series, they, therefore, are not attacked, but remain as metals, forming the sludge. The oxides form during drying.

Preparing Anode Sludge for Treatment.

The sludge, on being received at the "silver house," is first poured through a copper screen (about 10-20 mesh) in order to remove any nodules, or crystals, of metal. The screenings are washed and returned to the refinery, and melted down with the next lot of metal to be cast into anodes.

In case the material is copper anode sludge, after it is screened, it is poured upon a filter of muslin and filtered by suction or put through a filter-press to remove excess of electrolyte, which is returned to the refinery. The filtered sludge is washed with water, well drained, placed in sheet steel pans and dried in an oven. When dry, it is sampled and the sample sent to the assay office. The dry sludge is then treated for metal content by one of the following methods.

If the material is lead anode sludge, after it is screened, it is well washed with hot water by decantation, then filtered by suction on a muslin filter. The solution and wash water is evaporated in copper pans to about 1.15 Sp.Gr. and returned to the refinery. The washed sludge is placed in pans and dried and partially oxidized in an oven, and finally treated by one of the following methods.

Treatments.

A. Direct Cupellation: (Method followed when copper refinery is run in connection with the Parkes' or Pattinson's de-silverizing plant). Used for treatment of copper anode sludge: The dry sludge placed in heavy paper bags holding 10 to 15 lb. and charged on the molten metal in a cupelling furnace. The Cu, Pb and Bi are oxidized and collect with the litharge on the surface of the metal as scoria, which is skimmed off and sent to smelting furnace, where added along with regular furnace charge. The As, Sb, S, Se & Te are oxidized and volatilized, and partially enter the scoria. Difficult to remove all of the Se & Te from the resulting Doré bullion.

The Doré bullion, assaying 900 + fine, is cast into suitable slabs, and parted either with boiling concentrated H_2SO_4, or else by electrolysis.

B. Fusion Methods.—For treatment of copper and lead anode sludge: The washed and dried sludge is mixed with a flux of soda ash (crude Na_2CO_3) and small amount of KNO_3 and smelted in a basic lined (magnesite) reverberatory furnace, producing a slag which contains the Sb, As, Se, Te, S and most of the Cu and Bi. The slag is sent to the ore smelting furnace and added along with regular charge. The bullion remaining on the hearth is cupelled to Doré bullion (alloy of Ag, Au, Cu, etc.), and the resulting scoria of Bi_2O_3, PbO, Cu_2O, etc., is skimmed and sent to the smelting furnace. The Doré, assaying 900 to 975 fine, is cast into suitable plates and parted either with hot conc. H_2SO_4 or by electrolysis.

C. Fractional Scorification in Basic Lined Reverberatory Furnaces.—For sludge which contains small amount of copper, such as electrolytic lead refining sludge. The sludge, after washing and drying, is scorified (oxidizing atmosphere above melting point) in the "antimony-slagging" furnace till the arsenic and antimony are mostly removed. The scoria is skimmed and subsequently treated in another furnace for the production of hard-lead. The metal is withdrawn from the scorifying furnace and placed in a second basic lined furnace and rescorified for the removal of the bismuth as Bi_2O_3. When most of the bismuth is scorified, the scoria is completely skimmed and subsequently treated for the bismuth by chemical leaching methods. The metal remaining in the furnace is withdrawn and finally rescorified in a third basic lined furnace to remove the remaining lead and most of the copper, leaving Doré bullion. The lead-copper oxide scoria is smelted along with the regular charges in a lead blast furnace, recovering the copper as matte and the lead as base bullion. The Doré bullion is cast into thin plates, and parted, either with concentrated H_2SO_4 or electrolytically.
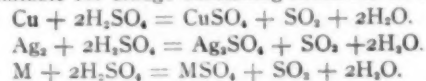
When high grade bullion is treated by either of the above methods, precautions must be taken to *collect the flue dust*. This is done by passing the gases through sheet-steel flues 100 to 350 ft. long, then through a series of dust chambers and finally catching the finely suspended particles by filtering the gases through long flannel bags, suction draft being used, or else passing into spray-towers, or under water.

The flue-dust from these furnaces is returned to the smelting furnaces, or else treated the same as sludge. If much As_2O_3 present in the flue dust, it is recovered by subliming twice in a muffle furnace or in a reverberatory furnace using coke fuel. The residue is mixed with next treatment of sludge.

In some plants, where the treatment is carried out by either A. or C., the scoria removed during the latter part of the treatment runs high in silver, so is kept separate from that which
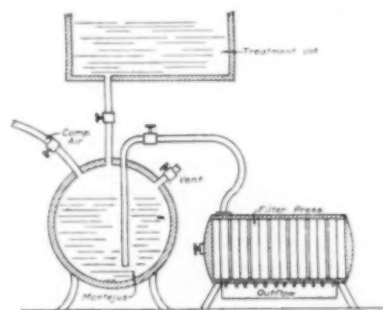
is produced during the first part of the operation. The first scorias, in this case, are sent to the smelting furnace, and that which runs high in silver is added along with the next charge of sludge to the furnace, and its silver content collected.

D. Direct Parting with Boiling Conc. H_2SO_4.—Not used at present, as much acid is lost due to formation of SO_2. (Not suitable for sludge containing much As or Sb).

$$Cu + 2H_2SO_4 = CuSO_4 + SO_2 + 2H_2O.$$
$$Ag_2 + 2H_2SO_4 = Ag_2SO_4 + SO_2 + 2H_2O.$$
$$M + 2H_2SO_4 = MSO_4 + SO_2 + 2H_2O.$$

This treatment is similar to the parting of Doré bullion with boiling conc. H_2SO_4 in iron kettles.

E. Mix the washed and dried sludge with a calculated amount of conc. sulphuric acid (sufficient to dissolve the copper, antimony, etc., but not enough to dissolve the silver), forming a thick mud. It is then heated in iron kettles till SO_2 is expelled, and silver begins to dissolve. A small amount of fresh sludge is added to precipitate the silver, then the residue is leached with water to remove the sulphates. The solution is treated by the usual method of crystallization, or by passing over iron, for recovery of the metals. The residue after leaching is scorified to Doré bullion, which is finally parted. The scoria is smelted in a lead smelting furnace along with regular charge.

FILTER PRESS AND MONTEJUS.

The cement metal, formed by precipitation of the sulphate solutions with iron, is smelted for its metal content.

F. Treatment with Hot Dil. H_2SO_4, Air and KNO_3.—The washed sludge is placed in a lead lined vat containing dilute H_2SO_4 (1 acid : 2 + water), and heated to boiling, and agitated by means of injected steam and air, and hard-lead stirrers. A small amount of KNO_3 (saltpeter) is added at intervals in order to assist in the oxidation and more rapid solution of the copper, etc. This treatment lasts from 20 to 30 hours, during which time most all the copper and some Ag is dissolved, and Bi and Sb changed to basic sulphates, and Pb to $PbSO_4$. The Ag in solution is precipitated by suspending strips of copper in the tank at the end of treatment, and collecting it with the residue of Au, Ag, $PbSO_4$, BiO_2, SO_4, $Sb_2O_2SO_4$, etc., or else by adding a small amount of fresh sludge.

The Cabell Whitehead process is a modification of the above treatment, using Ag_2SO_4 residues from the parting kettles in place of KNO_3 to get the Cu, Bi, etc., in solution. Process formerly used at plants where the Doré was parted with boiling conc H_2SO_4.

Reaction.—$Cu + Ag_2SO_4 = CuSO_4 + 2Ag.$
$$2Bi + 3Ag_2SO_4 = Bi_2(SO_4)_3 + 6Ag.$$
$$M + Ag_2SO_4 = MSO_4 + 2Ag.$$
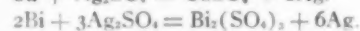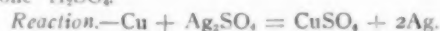
In case of Bi and Sb, their sulphates are decomposed, forming insoluble basic sulphates ($Bi_2O_3 . SO_4$ and $Sb_2O_3 . SO_4$).

The use of the Ag_2SO_4 residue from the H_2SO_4 parting kettle reduces the time of treatment six to eight hours. These residues consist principally of anhydrous Ag_2SO_4 and $CuSO_4$, which form on the side of the kettle during the parting of Doré bullion.

The vats in which the treatments are carried out are made of heavy pine boards lined with sheet lead. Dilute H_2SO_4 solutions do not dissolve metallic lead after it is coated with a thin layer of $PbSO_4$.

The vats used for evaporating or for boiling *dilute sulphuric acid* solutions are made of hard sheet lead (containing 4 to 6 per cent Sb), the heating being done by means of hard lead coils through which steam is passing.

Tree of Treatment of Anode Sludge with Dilute H_2SO_4 (F).

1. *Use of KNO_3 as Oxidizing Agent.—*

Anode Sludge containing Cu, Pb, Sb, Bi, As, Ag, Au, Se, Te, S.

Boil for 20-24 hours with dil. H_2SO_4 ($1A : 2+w$); boil and agitate with steam and air. Add, about every hour, small amount KNO_3 (total 5 to 10 per cent wt. of sludge).

Precipitate the dissolved silver on sheet copper, or with fresh sludge.

Solution of $CuSO_4$: $FeSO_4$, $Sb_2(SO_4)_3$: $Bi_2(SO_4)_3$, As_2O_3 (H_2O).

Residue of (Cu) : $PbSO_4$: $Sb_2O_3(SO_4)Bi_2O_3(SO_4)$, Ag, Au, Se, Te, S.

Same treatment from now on given under *"Final Treatment of Solutions and Residues."*

2. *Cabell Whitehead Process—Use of Ag_2SO_4 as Oxidizing Reagent. (F).*

Anode sludge cont'g Cu, Pb, Sb, Bi, As, Ag, Au, Se, Te, S.

Boil for 14-16 hours with dil. H_2SO_4 ($1A : 2+w$); boil and agitate with steam and air. Toward end hang several strips of copper in tank as precaution to precipitate Ag, if any in solution.

Solution of $CuSO_4$, $FeSO_4$, $Sb_2(SO_4)_3$, $Bi_2(SO_4)_3$, As O (HO)$_3$. Treat as under *"Final Treatment of Solutions and Residues."*

Residue of Cu, $PbSO_4$, $Sb_2O_3.SO_4$, $Bi_2O_3.SO_4$, Ag, Au, Se, Te, S.

Boil for 2 to 3 hours with fresh lot of dilute H_2SO_4, ($1A : 2+w$) and Ag_2SO_4. Residue from the parting kettle. At end of treatment precipitate the remaining Ag_2SO_4 with strips of copper, or small amount of fresh sludge.

Solution of $CuSO_4$, which crystallize, or send to copper refinery.

Residue of (Cu), $PbSO_4$, $Sb_2O_3SO_4$, $Bi_2O_3SO_4$, Ag, Au, Se, Te, S.

Same treatment from now as given below.

Final Treatment of Solutions and Residues.

Solution of $CuSO_4$, $FeSO_4$, $Sb_2(SO_4)_3$, $Bi_2(SO_4)_3$, As_2O_3 (H_2O)$_3$.

Residue of (Cu), $PbSO_4$, $Sb_2O_2SO_4$: $Bi_2O_2SO_4$, Ag, Au, Se, Te, S.

Evap. to crystallization in lead pans, heated by steam coils.

Wash by decantation, filter and dry.

$CuSO_4 . 5H_2O$.

market

Mother liquor of $CuSO_4$, etc.

Smelt with 20 per cent wt. flux of soda ash and KNO_3 in basic lined Reverb. Furn.

Evaporate to crystallization, getting second crop of crude $CuSO_4.5H_2O$.

Slag of Sb, Te, Se, S, (Cu_2O, $PbSO_4$, Bi_2O_3) and 1 per cent to 2 per cent Ag.

Impure $CuSO_4$, $5H_2O$, dissolve in water and recryst.

Evaporate to 66° B (SO_3 fumes) cool.

Mother Liquors.

Add to charge of smelting furnace.

Residue, CuO, As_2O_3, Sb_2O_4, BiO, FeO (FeSO)

Cone. H_2SO_4 (crude), use for next lot.

Base bullion, Ag, Au, Pb, Bi, Cu, trace Se and Te.

May be treated for As_2O_3, Sb and Bi, or else send to smelting furnace.

Cupel, to Dore bullion (90 + Ag, 5 + Au and Cu, Se, Te).

Litherage cont'g Cu, Bi (Sb, Se, Te). Sent to Pb smelting furnace.

Cast into plates and part with boiling H_2SO_4, or by electrolysis.

The boiling of *concentrated sulphuric acid* is done in cast iron or cast steel pans, heated by flame underneath.

The *filter presses* used for filtering sulphuric acid, or sulphate solutions, are made of hard lead (containing 8 per cent Sb), the filtering medium being closely woven cloth. The solutions are forced through the filters by either pumps or montejus, which are made of iron lined with hard lead. A montejus is an air-tight siphon vessel, used for lifting liquids by means of compressed air. The solutions are run into the montejus by gravity and afterward forced out through a pipe extending to the bottom, the supply pipe being closed. This apparatus is more suitable than a pump, as it has no working parts which are exposed to the acid solutions.

The *crystallizing vats* for $CuSO_4. 5H_2O$ are made of wood and lined with lead. Strips of lead are suspended in the vat and form the nucleus for the crystals of $CuSO_4. 5H_2O$. The hot saturated solutions are run into the vat and allowed to cool, giving a crop of crystals, which, if pure, are collected, washed, dried and stored. If the crystals are impure they are dissolved and recrystallized. The final mother liquors are passed over scrap iron to recover their copper content and the solutions run to waste. The cement copper is added to the copper furnace.

The *scorifying furnaces,* in which the acid-treated sludge is brought to Doré bullion, have hearths 5.5 to 5 ft. wide and 6 to 8 ft. long, capacity of 60,000 to 150,000 Troy ounces of Doré per charge. The hearths are made of either magnesite or chromite bricks, which are not corroded by the scoria. The furnaces are usually fired by oil and blast. Ordinarily two charges of Doré bullion are completed and cast per week.

A comparatively cheap, but still very satisfactory, material for making hearths of scorifying and cupelling furnaces is made of dolomite concrete, which is prepared by mixing about four parts of crushed dolomite and one part Portland cement. This is thoroughly mixed with sufficient water to give a good "setting" mass, then tamped into place, allowed to "set," and thoroughly dried by a fire of gradually increasing intensity.

The workmen, before they are allowed to leave the "silver house," are required to change their clothes and wash. The waste wash water from the sinks is run into settling tanks where the slimes collect. The slime is removed from the settling tank at stated intervals and is treated along with the anode sludge. The old cloths are burnt, and the ash treated as sludge.

Notes on Chemistry and Metallurgy in Great Britain.

(Continuation, from our last issue, of the report of the Iron and Steel Institute.)

Mr. J. E. Stead's paper on "Notes on the Welding-Up of Blowholes and Cavities in Steel Ingots" was next read.

Professor Arnold spoke of the remarkable effect of a small proportion of aluminium in preventing blowholes. For many years it had been an annual experiment at Sheffield University to cast two ingots of the same weight, one without aluminium and the other with 0.01 per cent of aluminium, and the latter was entirely free from blowholes.

Mr. S. H. Ridsdale said that hollowness in forgings was often caused by working at too low a temperature in order to save re-heating. He had seen blooms which showed splits in the forged ends and middle, but were proved, by polishing and etching, to be quite sound in the unforged parts which, under proper conditions, yielded thoroughly sound forgings. He could not concur in Mr. Stead's statement that blowholes were least objectionable when well within the surface. Rails with a very solid exterior almost always had a spongy interior, while if the surface were spongy and shelly the inside was almost invariably sound.

Dr. W. Rosenhain remarked that the fact of a hole not reopening was not proof that it had been welded up. The time during which pressure was exerted on the steel was an important factor. There was an inward diffusion of air apart from pressure. He could not accept as proved the author's statement that a coating of iron oxide on the interior of a blowhole was a bar to welding, because he had evidence that where the skin of oxide could be broken through welding might occur.

Mr. W. H. Ellis referred to a case of an internal cavity in a large 30-ton fluid-compressed ingot. Apparently when the walls attained a certain thickness a cavity formed. He had forged down such an ingot, cut it into lengths and found that it welded satisfactorily. He believed that the reason was that the interior of the cavity was free from oxide.

A paper by Messrs. E. F. Law, W. H. Merett and W. Pollard Digby, "Some Studies of Welds," followed. Opening the discussion, Mr. A. E. Tucker said he was anxious that it should not go forth that defects such as were detailed in the paper were insuperable or even serious in practice. Gas welding had already been largely adopted with complete practical success. He produced a sample to illustrate what might be done by cutting steel with the oxy-hydrogen flame in discovering defects in the body of the metal which seemed undiscoverable in any other way. The two surfaces were very interesting. The first, cut with pure oxygen, was but little inferior to one obtained with a rough milling cutter, while the second, cut with a 96 per cent oxygen, was ragged and heavily oxidized.

Professor Turner said the microscope afforded an easy means of determining which process was best adapted for the production of a particular kind of welding. There were defective weldings by the acetylene process or by the resistance process, but that only meant that the operations were not properly carried out. He believed that those welds by modern processes when properly effected were perfectly satisfactory, as was shown by the tensile strength and working properties. At the same time the authors were doing a public service in drawing attention to certain possibilities of failure.

Sir Thomas Wrightson remarked that only certain metals could be welded. The melting of two metals together was not the same thing as welding. Many years ago he had made experiments at the Mint with Roberts-Austen which demonstrated that in the case of iron there was a distinct fall of temperature at the moment of solidification, when the pressure came on, and not an evolution of heat. He mentioned this as an explanation of the welding properties of certain metals.

Mr. H. Le Chatelier (Bessemer Medallist) said the inevitable defects of welds were all known and it would never be possible to entirely obviate them. The most pressing problem of the day was to find some means of ascertaining the quality of a weld without at the same time destroying it. Possibly the electric, magnetic or acoustic properties of the material might afford an index of even internal defects.

Dr. J. E. Stead quite agreed with the authors that it was possible to tell the method of welding by the microscope. His own experience had often confirmed the fact.

The following papers were discussed together: "The Action of Aqueous Solutions of Single and Mixed Electrolytes Upon Iron," by Dr. J. Newton Friend; "The Corrosion of Metals," by Mr. P. Longmuir, and "The Influence of Impurities on the Corrosion of Iron," by Mr. J. Cobb.

Professor Arnold remarked that one interesting point in Mr. Longmuir's paper was the mechanical deterioration of metals as a consequence of corrosion by sulphurous or sulphuric acids. This was a matter deserving attention.

Mr. E. H. Saniter fully appreciated the value of Mr. Longmuir's insistence on the importance of studying the effects of atmospheric corrosion as well as local corrosion set up by electrolytic action. He considered that in many cases laboratory tests on corrosion were quite misleading.

Prof. H. C. H. Carpenter read a paper on "The Growth of Cast Irons after Repeated Heatings." This was a continuation of his paper on the same subject last year.

Mr. J. E. Stead opened the discussion and said he had examined a stove pipe which had been heated for several years at a constant temperature of about 750° C., and the microscopical, physical and chemical results obtained led to the important conclusions which might modify some of Dr. Carpenter's tentative conclusions. Mr. Stead exhibited micrographs of the structure of the pipe before and after heating. Gas penetrated the metal very easily, but the pipe had not been heated above 750° C. and all the pearlite in the area of ferrite was globulized, while the central parts of the dark areas consisted of laminæ of graphite. When broken the cleavage was through the graphite, and, although about 25 per cent of slag was present, the fracture of the pipe was just the same as that of the original pig. He was of opinion that the shape of the sample was a factor in the longitudinal expansion. If the bars had been round different results would have ensued. Dr. Carpenter naturally assumed that silicon steel was more easily oxidized, but steel containing no silicon had been proved to oxidize to the greater extent.

Professor Arnold said that Dr. Carpenter classed as steel a material which was something between steel and pig iron. The material specified with 2.5 per cent of carbon was a special steel which commanded a high price in Sheffield. He suggested that the material Dr. Carpenter proposed to substitute was not cast iron, but a hard and probably brittle steel.

Mr. Saniter was not able to accept Table V. An expansion of 11 per cent in vacuo of one alloy could not be altogether attributed to dissolved gases.

Professor Turner highly appreciated Dr. Carpenter's work and congratulated him on being able to indicate the composition of an alloy which would not expand on heating. An alloy containing 2.66 per cent of carbon with 1.5 per cent of manganese should run quite well. He did not agree with Professor Arnold's remarks as to Sheffield steel, Professor Arnold called it steel, but he called it cast iron for the addition of a little phosphorus gave it almost exactly the composition of pig. The irons experimented on were very complex materials and could be heated to such temperatures as would cause absorption of the pearlite and diffusion and absorption of phosphorus. Some volume changes seemed to result from alteration in the silicon-iron-carbon compounds.

Mr. Edwards considered the author's explanation of growth the only one reconcilable with observed facts. Oxidation could not take place under those conditions. There might be alloys of various compositions evolving different gases, but growth might be due to the nature of the gas and not to its volume.

Mr. Adamson agreed in the author's view of the effect of manganese and phosphorus on growth. He had proved its correctness several years back with castings required to stand high temperatures in working. Dr. Carpenter was, doubtless, right in his contention as to the influence of sulphur under those conditions of test, but when sulphur in the form of acid was brought into contact with iron castings, such as gas retorts, a deposit resembling pyrites resulted, with destruction of the iron. Consequently the less sulphur such iron contained the longer was the working life of the casting. There has not yet been brought to light any proof that such destructive action of sulphur was connected with the growth of cast iron.

In the discussion on the paper by Professor A. McWilliam and E. J. Barnes, "The Influence of 0.2 per cent. Vanadium on Steels of Varying Carbon Content," Mr. J. Kent Smith said vanadium toughened steel by scavenging from it the suboxides and nitrides. The products were fusible and formed with the vanadium a solid solution in ferrite. The great strength of vanadium steel was due to the formation of complex carbides. With regard to cast iron he was able to confidently assert that the scavenging action followed on the use of vanadium. Notable advantages of vanadium cast-iron were high resistance to abrasion and freedom from porosity.

The Metallurgical Future of India.

The development of the natural resources of India and their resulting industries is, and must necessarily be, more a matter of judicious government encouragement than in countries where private enterprise is more in accordance with political conditions. The existence of various intricate systems of established rights renders interference with existing customs a process of grave importance, and one which cannot always be entrusted to a commercial body with more prospect of permanent success than can the public safety be handed over to the care of the half-educated amateur imported politician.

Previous to 1899 the complications caused by a fluctuating exchange made the extension of mining concessions impracticable, the financial return being too uncertain to form a steady inducement to private enterprise. The regulation of the coinage question which came into effect in that year, however, permitted the Indian government to take the matter up and frame regulations which offered more scope for private enterprise.

Some idea of local conditions, and the care which must be exercised in making alteration, however slight, in the usages of centuries, can be gathered from the comprehensive and instructive paper recently read before the Indian section of the Society of Arts by Prof. Sir Thomas H. Holland, formerly director of the Geological Survey of India. Amply qualified as he is to deal with the question from a scientific point of view, Sir Thomas is compelled by his subject to treat occasionally of the social or political aspect of the matter, as in the paragraph on relative safety of European and Indian mines. The superiority of the Bengal mines he attributes, in a great measure, to the fact that the Bengal collier retains, so far, a sense of discipline "that has now been all but destroyed by our education authorities in this country." Possibly a consequence of controlling instruction by those who are most in need of it.

While so many of the finest intelligences are thus engaged in directing the affairs of a portion of the empire which does not govern itself automatically, there is every hope that the mineral resources of the country will develop in conjunction with its other activities.

Taking the products in the order given them by Sir Thomas Holland, coal naturally leads in importance. We learn that during the past 25 years the production of coal has risen from 1¼ million to 12 million tons a year, and still increases, the best being that obtained from the Giridih field. This field is estimated to contain about 70 million tons. The author of the paper considers that a considerable impetus could be given to the industry in Central India by the construction of a broad gage railway extension from Nagpur to Itaisi, which would tap also a rich cotton-producing country, as well as the principal manganese mines.

The coal fields of Assam are at present worked principally for the local market, expense of transport cutting them off from Calcutta, as well as the quality of the coal, which is friable and liable to spontaneous combustion, and thus not easily carried for long distances.

The development of this industry depends very largely upon railway extension, both as providing a market and a means of transport. The available foreign markets, says this paper, are comparatively small, the principal consumption being due to locomotive wants. It seems possible, however, that a market for shipping might be cultivated where the quality of coal permits. At present, and since 1900, the exports of coal largely exceed the imports, which latter have been generally limited to small quantities to meet local and temporary variations in the demand for special coals. The exports have, on an average, been over half a million tons per annum.

With regard to petroleum the difficulties due to ancient customs for a considerable time hindered the development. When

these were overcome the output increased rapidly, rising during the past ten years from 37¾ million gallons to over 233½ million gallons (1909). Certain of the fields, however, show signs of exhaustion, the Twingon, in Upper Burma, for instance. On the other hand, three promising properties remain to be developed in Burma, the Khodaung, Singu and one on the other side of the Irrawaddy.

At present the indigenous production is insufficient to meet the country's requirements in kerosene and petrol, of which 90 million gallons are imported annually.

The production of iron is at present low in the peninsula, only one company operating with favorable results (the Bengal Iron & Steel Company), although another (the Tata Iron & Steel Company) has now been floated with a capital of £1,500,000 sterling to exploit the hematite ore deposits of the Mombhanj State in Orissa. The new works will be within forty-five miles of the principal ore supplies and 180 miles from the collieries. The ore in the Mombhanj is rich, carrying over 60 per cent of iron, and exceptionally easy to work, standing in hills above the general plain. In a period when rich ores were in demand they might thus be worth the attention of European furnaces.

Data of other fields do not seem available, although iron is known to exist, in small quantities at any rate, throughout the peninsula.

Turning to minerals raised chiefly or entirely for export the author finds that gold is only mined in Mysore and at Hutti in the Nizam's dominions. The principal opening for development appears to be on the upper reaches of the Irrawaddy, where dredging operations for alluvial gold have been carried on for the past nine years. The production in 1909 was valued at £32,730.

Manganese is a flourishing industry and appears capable of considerable and rapid expansion whenever the market provides the necessary encouragement, as shown by the output in 1907, which expanded from 571,500 tons to 902,291 under this stimulus. The falling off in steel manufacture and lower prices have since caused the output to drop to 600,000 tons.

In mica the country has a valuable product, which only needs control by organized limited liability companies to become of first-class importance. Even under present circumstances the production is equal to 61.78 per cent of the quantity produced by the three principal mica-bearing countries—India, Canada and the United States. The present methods of winning are hampered by the disadvantages consequent on divided effort and may be regarded as mere prospecting. Numerous deposits still lie untouched.

The further development of India in this connection may bring forward copper as a potential product. Copper ores are known to occur, and, in the past, have been considerably worked in the Nellore district of the Madras Presidency, in Rajputana, in Chota Nagpur, and at various places along the lower Himalayas from Kulu in the northwest to Sikkim and Bhutan in the east. Operations are now in progress by the Cape Copper Company to test the probable value of a copper-bearing country in Bengal. Prospecting operations are also in progress in the Darjeeling district and adjoining parts of the Sikkim State.

Lead, zinc and silver ores are known to exist and are being prospected on a considerable scale in upper Burma. Pitchblende and wolfram are also found, the latter in some quantity. Corundum is widely distributed in various forms, but it is possible that, in face of competition from natural and artificial abrasives in the countries of principal consumption, it would not pay at present to develop the Indian mineral for export to Europe and America.

The principal requisite for further development in the production of minerals in this country appears to be means of transport, a question which will have the consideration of those interested. The opening up of any particular field is almost invariably stimulated, in the first instance, by a brisk market. Unfortunately the period of high prices is very apt to terminate just as the preparations for opening up a new mine are ended, and it is in this connection that mercantile foresight becomes as valuable to the country as to the individual.

The continually urged objection to investment of political unrest the author does not consider grave. The results are temporarily sensational without having any widespread or permanent significance, like the political speeches of some home enthusiasts, while they serve to rouse to a stricter sense of duty those whose patriotism is not so case-hardened as to render them incapable of belief in sedition.

Recent Metallurgical and Chemical Patents.

Gold and Silver.

Silica Sponge Filters.—In two recent United States patents Mr. E. G. Spilsbury, of New York City, describes the use of a commercial product known as silica sponge for the purpose of filtering liquids or gases.

The treatment of gold and silver ore by the cyanide process involves methods of agitation and filtration which the inventor believes can be advantageously performed by the medium of a porous septum of mineral composition, such as silica sponge.
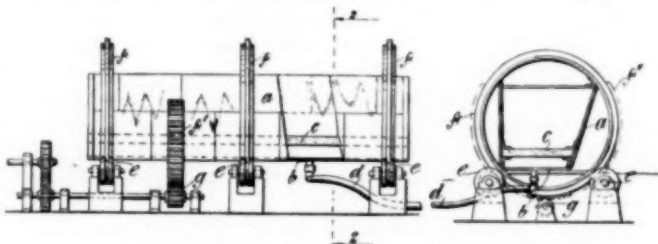

FIG. 1—SILICA SPONGE FILTER IN CYANIDE PROCESS.

As shown in Fig. 1, he proposes to place a septum of porous material c in a revolvable tank a, operated as shown in the figure. Beneath the septum is the bottom of the tank b, which is connected with a source of vacuum, compressed air and water by means of the flexible tube d.

The process of treating a charge of ore would be as follows: The ore having been crushed to the requisite degree of fineness and mixed with the cyanide solution would be charged into the tank a. For agitation of the pump compressed air would be admitted through the tube d, and when that operation was completed the air would be disconnected and vacuum applied through the same tube. This would draw off the pregnant solution from the ore, which could then be washed by water introduced through the tube d and filtered again as before. In this way the porous septum of silica sponge would act as a means of securing thorough agitation and aeration as well as filtration. When it is desired to discharge the filter cake the tank is rotated and the cake allowed to fall off by gravity, assisted by compressed air or water. (994,281, June 6, 1911.)

In applying the silica sponge to the filtration of gases, such as blast furnace gases from the smelting of lead and copper ores, the inventor proposes to use it much after the manner of using cotton and woolen bags in bag-houses. The mineral substance, however, will have the advantage that it will not be affected by the heat or acid nature of the fumes. The construction of a building for filtration of gases would be much the same as used in bag-houses, with a large chamber below and the filtering medium above, except that the bags would be replaced by a cell structure composed of a series of vertical cells alternately open and closed at the top, those cells which are closed at the top being open at the bottom, as shown in Fig. 2, which is a vertical section of the upper part of the filtering arrangement. The advantage of this arrangement is that the furnace gases will pass upward into those cells which are open at the bottom, filter through the porous walls of the

cell and escape to the air through the cells which are open at the top. When it is desired to clean the cells of the accumulated solid cake of fume, air can be forced in a reverse direction to the flow of furnace gas and cause the adhering cake to fall from the cell walls and down into the chamber beneath. This is an interesting modification of the present bag-house system and may have a bearing on the future operation of smelters in relation to agricultural interests which object to the discharge of noxious gases and fume. (994,282, June 6, 1911.)
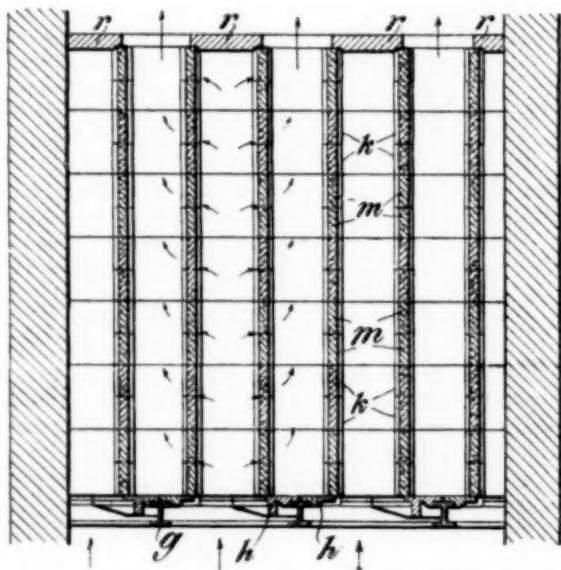

FIG. 2—FILTRATION OF GASES WITH SILICA SPONGE FILTER.

Rotary Filter.—A modification of the type of revolving filter commonly used in the United States has been patented by MR. ALEXANDER J. ARBUCKLE, of Johannesburg, Transvaal, South Africa. In the Oliver and Portland revolving filters used in this country the filtering medium is on the outside of the drum only, but in the Arbuckle filter it is proposed to make use of both the outside and inside surfaces of the drum for filtering purposes. The drum is divided into sections which, by means of a three-port valve located in the hub of the drum, are brought successively into connection with a vacuum for forming the cake, a wash and vacuum for removing the retained values and compressed air for discharging the cake. For this purpose three non-communicating spaces are provided in the hub, one for each process mentioned. Consequently, as the drum revolves in a tank of ore pulp, each section successively forms a cake on both its surfaces, which are later washed and discharged. (994,235, June 6, 1911.)

Air Lift Agitator.—In Fig. 3 is shown an improvement in air lift agitators for use in the cyanide process, patented by LEON P. HILLS of Tuolumne, Cal. The object of the invention is to provide means for bringing all the pulp in a tank into circulation, even after the ore may have settled and packed around the opening of the lift, and for securing a quiet overflow of clear solution in case of continuous treatment. The operation of the tank will be understood by reference to the figure, in which 33 is a launder and 34 a pipe for continuously introducing pulp and solution respectively. An air pipe 19 extends downward through the inner central tube, terminating in a U at the bottom. The central lift 12 is surrounded by a movable tube 20 which is operated by the hand wheel 25, this outer movable tube being shorter than the inner tube. A circular baffle 30 is placed in the upper part of the tank, as shown. The tank being full of a mixture to be agitated, the outer tube 20 is lowered into contact with the walls of the tank at the bottom, and air is admitted at the bottom of the inner tube. This will cause a circulation of the nearly clear

solution, which will rise through the inner tube, spill over the apron 14 at the top and return through the annular space between inner and outer tubes, as shown at 29. When this circulation is thoroughly established the outer tube is raised gradually to admit the solid pulp at the bottom of the tank into the inner tube, and in this way the entire mass will ultimately come into circulation. The baffle 30 acts as a guide to prevent the pulp from flowing to the outer edge of the tank and directs it downward, thus preserving a quiet zone between the baffle and the wall of the tank where solution more or less free from pulp
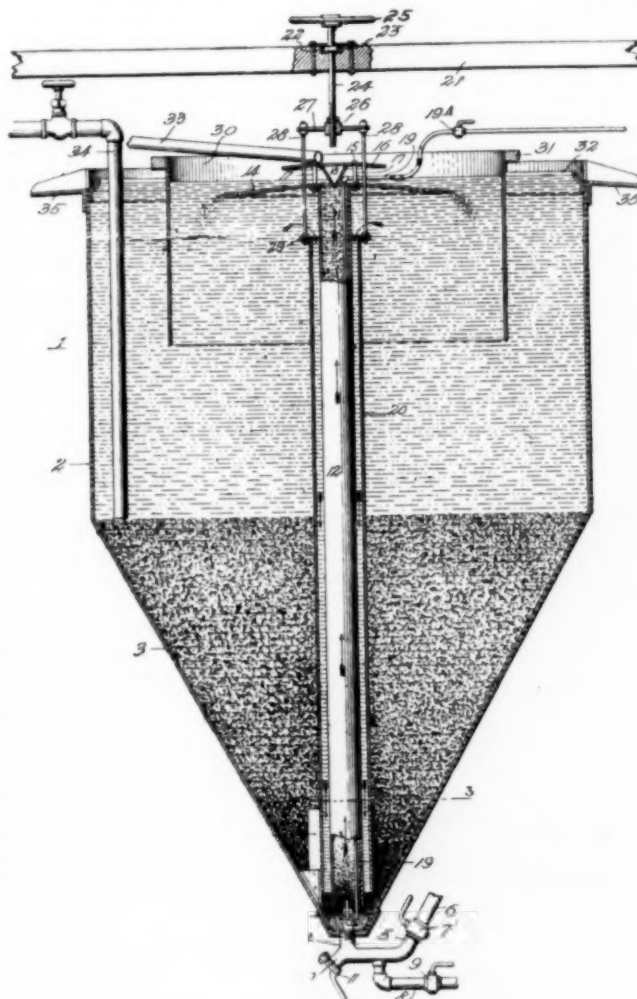

FIG. 3—AGITATOR.

can be drawn off through the circular launder 35. At the bottom of the tank a valve is provided for sluicing out the settled pulp when its values have been extracted by agitation. (994,-679, June 6, 1911.)

Lead and Copper.

Fume Neutralization.—The system of fume neutralization as used by the U. S. Smelting Company at Salt Lake City, Utah, is the subject of a patent issued to MR. CHARLES B. SPRAGUE, of Salt Lake City.

Owing to the acid nature of the gases given off in the roasting of ores in converter and reverberatory roasting, it is impossible to filter those gases through bags in the manner customary with lead smelters. A system of neutralization of the sulphuric acid contained has been found possible, after which the gases can be filtered and discharged into the air, free from noxious substances, either solid or gaseous.

For neutralizing the acid in gases from converter roasters of the Huntington-Heberlein type, which are not as highly charged with sulphuric acid as those from reverberatory roasters, zinc oxide has been found efficient. Powdered lime, how-

ever, is preferred for neutralizing the gases from reverberatory roasters. In operation a furnace is built adjacent to a flue leading from the converter roasters, in which a mixture of zinc ore and coal slack is burned under a low blast. The zinc oxide produced is conducted into the roaster flue, where it mingles with the roaster gases and neutralizes the acid contained. When powdered lime is used, as on reverberatory roaster gases, it is introduced by means of a fan and eccentric feeder. The gases pass through the fan, and the feeder is set to deliver the proper quantity of lime into the gas.

Flues leading from both converter and reverberatory roasters may be united after the separate treatments have been applied and the combined gases filtered through bags as usual, the corrosive action of the gases having been overcome. (992,-391, May 16, 1911.)

Roasting Furnace.—Improvements in the manner of attaching and detaching rabbles and rabble arms in ore roasters of the superimposed-hearth type have been made by MR.
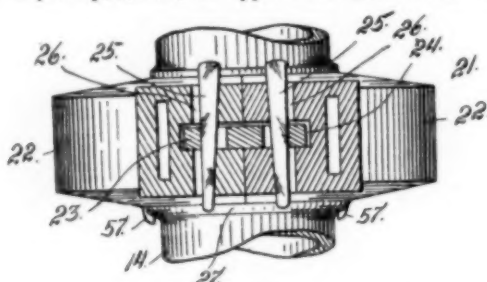

FIG. 4—DETAILS OF ROASTING FURNACE.

CHAUNCEY E. DEWEY, of Denver, Col. Figs. 4, 5 and 6 illustrate the invention. Referring to Fig. 4, the vertical central tube which carries the rabble arms in a furnace of the type described is represented by 14. The hub which carries the arms is composed of two twin members 22 which are connected by locking plates 23. As shown in Fig. 5, the hub is equipped with sockets adapted to receive the cone-shaped extremities of the rabble arms, which are fastened in the sockets by bayonet
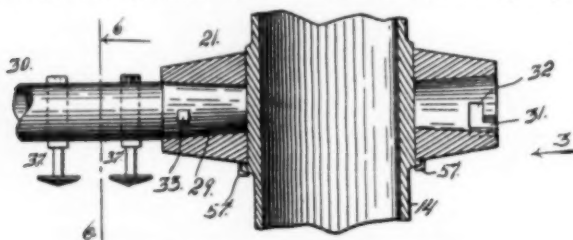

FIG. 5—DETAILS OF ROASTING FURNACE.

joints, so that when the furnace is in operation the movement of the rabble through the ore tends to hold it tightly in place. The rabbles themselves, which are shown in Fig. 6, are adapted for easy attachment or detachment. They are held in place by pins 34, which engage the perforations 35 made in the upper curved end of the rabble or plow. By means of the perforated lug 40 any plow may be removed from the rabble arm when necessary.

Alloys.

Ferrovanadium, Molybdenum and Tungsten.—The difficulty of preparing the ferro alloys of vanadium, tungsten and molybdenum usually has been that the reducing agent used remained in the resulting alloy in too large quantity. MR. GIDEON BOERICKE, of Philadelphia, Pa., has patented a process of reducing these alloys, which he claims eliminates the trouble experienced before. In carrying out his invention he uses a suitable form of the metals, such as vanadate of iron, which he mixes with manganese metal or alloy and a flux such as common sand. When the process of reduction is completed the products are ferrovanadium and a slag composed chiefly of the oxides of silicon and manganese. (993,338, 994,055, May 30, 1911.)

Purification of Magnesium and Alloys.—In the ordinary manufacture and refining of magnesium and its alloys more or less metallic chloride is retained in the metal, which subsequently has a deleterious action and causes oxidation of the metal in moist air. BERTHOLD HOFFMANN and ROBERT SUCHY, of Griesheim-on-the-Main, Germany, have discovered that it is possible to readily and completely free magnesium and its

FIG. 6—DETAILS OF ROASTING FURNACE.

alloys from this chloride by passing through the molten metal hydrogen or other reducing gas, which will drive off all traces of chloride at temperatures which are but little higher than the melting point of the chloride.

The process may be carried out as follows: About 8 kg of the metal are charged into a crucible fitted with a pipe through which hydrogen gas may be passed to the bottom. The current of gas is passed through the fluid metal for five or six hours at a temperature of 800° to 900° C. (993,-373, May 30, 1911.)
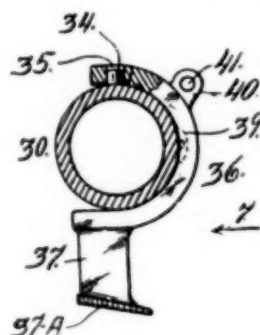
Pyrophoric Alloy.—Metallic alloys which give sparks when scratched with a hard substance have hitherto been made by fusing together the metals of the rare earths, especially cerium, with heavy metals like iron. ADOLF HUBER, of Berlin, Germany, has discovered that aluminium can be substituted for iron and that the alloy thus resulting must contain not more than 50 per cent of the aluminium or other earthy metal. When the percentage of the cerium metals falls below 50 per cent the property of spontaneous ignition of the separated particles almost totally disappears. The preferable percentage of aluminium to be added is 12 per cent, at which proportion the alloy can be cast without disintegrating on cooling. The lower specific gravity of the aluminum-cerium alloy is an advantage, as is also the lower melting point. (993,998, May 30, 1911.)

Electroplating.

Cleaning and Polishing Silver Plate.—Silver plate is cleaned and polished, according to Mr. A. M. KOHLER, of Brixton, London, England, by immersing in a sodium hydroxide solution in presence of aluminium or an aluminium alloy. When the articles are immersed in the solution and are in contact with aluminium, galvanic action is set up, the aluminium acting as anode and the silver articles as cathode. The anodic action is the formation of sodium hydroxide; the cathodic action is the reduction of the oxide which forms the tarnish on the silver to metal. Instead of metallic aluminium an alloy consisting of 80 parts aluminium, 10 barium, 8 calcium, 1 sodium and 1 silicon and impurities may be used. If the articles to be cleaned are greasy or very much soiled they may be immersed in a hot solution of Na_2CO_3 before placing in the solution as above described. (983,727, Feb. 7, 1911.)

Detinning.

Treatment of Old Tin Cans.—Preparatory to the removal of the tin from the iron old tin cans, etc., must be freed from everything that is neither tin nor iron. Dr. KARL GOLDSCHMIDT and Dr. JOSEPH WEBER, of Essen, Germany, subject the tin cans to the following treatment: First, removal of the adhering organic substances by treatment with a hot 3 per cent caustic soda solution for five to fifteen minutes. Second, removal of the solder and the caoutchouc ring (which in folding the edges of the tins is placed in the fold to make the same tight) by sudden exposure of the tin cans to a temperature of from 600° C. to 800° C., the solder being melted off in five to seven minutes and the caoutchouc being destroyed at the same time. Excessive heating must be avoided, as this would cause the tin to alloy with the iron. (13,123, June 28, 1910.)

SYNOPSIS.

Gold and Silver.

Cyaniding at the Waihi Mine.—In the *Transactions* of the Australasian Inst. Min. Eng., Mr. E. G. BANKS, metallurgist of the company, describes the method of treatment of an ore containing sulphides. All products are cyanided, but the sulphides are separated and treated separately from the balance of the ore.

Following rock breakers and stamps, the ore is ground in tube mills. The tubes are 18 ft. x 4 ft. 9 in., and have cast-iron liners in the shape of ribs 4 in. wide spaced at 15 in. centers. When new the ribs stand 3¼ in. above the body of the liner. They have a life of at least 18 months and represent a cost of a trifle over one cent per ton of stamp output. The consumption of pebbles is 2.6 lb. per ton of stamp output, representing a cost of about one and three-quarters cents per ton. The proportion of water to ore is kept at about 1 to 1, and each tube has a capacity of 110 to 120 tons of stamp product (10 mesh), although it grinds from 280 to 340 tons on account of the oversize returned from the sizing boxes.

Concentration follows tube milling, which is unusual practice, but justified by the author on the score that if the coarsest sulphide particles are removed it matters not if very fine sulphides escape to the cyanide plant. The sulphide concentrates are treated as follows: Grinding is accomplished in tube mills, and the coarsest of the tube product returned for further grinding. The ground pulp is thickened to 0.7 parts of solution to 1 of ore and agitated from 8 to 10 days by means of air in conical bottom tanks. Cyanide solution (NaCN) contains 0.45 per cent. of the salt, and the consumption is 16 lb. per ton, which represents 0.25 lb. per ton of original ore. The pulp is filter-pressed and the solution precipitated in zinc boxes.

The slime is thickened in V boxes and later by vacuum filtration, and is then mixed with solution of 1.2 per cent. KCN, and agitated by air in tall tanks for five days. The latest tanks built are of reinforced concrete 55 ft. high and 13 ft. diameter, erected at a cost of about $1,064, which is less than for similar sized steel tanks. No center pipe is used, slightly better results having been obtained without it. Gold is extracted in 24 to 30 hours, but a profitable extraction of silver continues up to five days. The consumption of cyanide is 1.66 lb. per ton of slime.

Vacuum filters are used to recover the pregnant solution, and zinc dust is used for precipitation. The extraction is 89 per cent. of the gold and 71 per cent. of the silver.

For several years the sulphuric acid process was used for refining, but was discarded two years ago in favor of the electrolytic method. The Doré bars run about 12 per cent. gold and 87 per cent. silver, and are separated in the stationary cathode type of cell. The output of 30 cells is about 5000 to 6000 oz. in two shifts (16 hours). The gold recovered runs 996 to 999 fine. The cost of refining is about 0.72 cents per oz. of Doré. The total cost of ore treatment is about $1.30 per ton.

Air Lift Agitation.—Writing on the subject of air lift agitation of slime pulp, Mr. ROBERT ALLEN presents some interesting data in the March, 1911, issue of the *Journal* of the Chemical, Metallurgical & Mining Society of South Africa. Referring to F. C. Brown's development of the so-called Pachuca tank, he says that the following points were found essential: The higher the tank compared with the diameter, the less the power required; the consumption of cyanide was less in tall than in shallow tanks owing to the fact that the necessary quantity of compressed air is less per ton of ore in the former than in the latter; although good agitation could be effected with slimy material in shallow tanks, yet as the coarseness or density of the material, or the specific gravity of the pulp increased, the ratio of height to diameter should be increased; the diameter of the central air lift pipe should be 1¼ in. for each foot of diameter of the tank.

On the score of excessive consumption of cyanide the author cites comparative results from the Loreto mill and the Hacienda San Francisco at Pachuca. In the former they use 30-ft. mechanically agitated tanks, with a charge of 57 tons in a thin pulp, while in the latter with Pachuca tanks agitated by air they work on charges of 112 tons of thick pulp. The cyanide consumption in the former mill is greater than in the latter and its power consumption is at least ten times as great.

Other experiments showed that Brown or Pachuca tanks consumed only 2.6 and 2.4 lb. cyanide on the same pulp on which mechanical agitators consumed 3.1 lb. cyanide per ton.

Lead.

Smelting in the Ore Hearth.—An interesting comparison of results obtained in smelting non-argentiferous galena in Scotch and Jumbo hearths at Granby, Mo., is given in the May, 1911, *Bulletin* of the A. I. M. E. by J. J. BROWN, JR., formerly connected with the Granby Mining & Smelting Company.

The ore hearth was at first a very wasteful means of reducing galena because no provision was made to recover the fume. Later this type of furnace was regarded with favor because it could be used to make a large percentage of fume or pigment, which was collected in the Lewis and Bartlett bag process. In fact, the production of metallic lead was kept down by hot blast, etc. Recently, however, a modification of the old Scotch hearth has been adopted for the reduction of lead, with a minor quantity of fume which is collected in bags, and slag which is reduced in blast furnaces.

The Granby company, however, having no blast furnace, was obliged to make as complete a recovery and leave as little lead in the slag as possible. About five years ago the system was as follows: The furnaces were water-backed Scotch hearths 2 ft. wide, using charcoal as fuel, and treating 14,000 of charge in about 14 hours, working two shifts of four men each. The men were required to convert 70 per cent. of the galena charged, 50 per cent. of the white fume, and 40 per cent. of the blue fume into pig lead. A premium of one cent per pound was given for better work than that required, but a similar penalty was charged for poorer work. Under this system about 85 per cent. of the lead in the charge was recovered, and about 2 per cent. went into blue slag, while the balance was recovered as fume and resmelted.

The author believed that better work could be done in Jumbo hearths and the result of his experiments was that all the Scotch hearths were remodeled into Jumbos. The percentages required of the operators were changed to 65 per cent. for galena, 60 per cent. for white fume, and 50 per cent. for blue fume contained in the charge, with the same premium or penalty as before. Cheaper fuel was used (bituminous coal instead of charcoal), and in a six months' run it was found that there was an advantage in favor of the Jumbo hearth of $0.6854 per ton of material smelted, or more than $1,200 per hearth per year.

The Jumbo hearths smelted a charge in less time than the Scotch hearths, the difference being about 4 to 5 hours per day. This is due to the larger fire area of the Jumbo hearth; the larger the hearth the more material can be smelted at one time provided the workmen can do the extra work entailed. A higher extraction was found on the 4-ft. than on the 5-ft. hearth, because the latter had too much fire to be worked advantageously. Pea-size ore is found to be the best.

A great objection to the ore hearth is that of small capacity. The author states that the only way to overcome this is to make the furnace mechanical in its operation instead of having it operated by hand. He states that according to designs of his own, in a patented furnace (888,582, May 26, 1908), the hearth may be from 10 ft. to 20 ft. wide, and worked by half the number of men.

Tin.

Analysis of Tin Ore.—Disregarding the fire assay for tin, which is regarded as inferior to volumetric methods, R. J.

Morgan gives his preferred method of estimating tin in an ore in the *Australian Mining Standard* of March 8, 1911. Six to eight grams of stick NaOH are heated in an iron crucible to expel moisture. About 5 grams of the finely powdered ore are added to the crucible after the NaOH has cooled and set. The crucible is then covered and gently heated to dull redness until fusion is complete. Allow the melt to cool, and then extract it with water in a casserole. Keep the washings as low as possible, but use enough weak HCl to wash the crucible and cover free from any adhering melt. Then cover the casserole and add HCl until effervescence ceases, and add 20 cc. strong HCl in excess. Heat to boiling and transfer to a suitable flask. There should be no undecomposed residue, save possibly some gelatinous silica.

Now add pure iron, in the shape of three horseshoe nails, to the solution and boil for 15 minutes. Filter through glass wool, washing with about 50 cc. water. To the filtrate add about 5 g. finely powdered pure antimony, heat to boiling and boil for 5 min. Cool as rapidly as possible in an atmosphere of CO_2. This can be done by dropping a small piece of marble in the flask, but a continuous source of CO_2 conducted through the flask is better. When cold the solution is titrated with standard iodine solution.

Zinc.

Flotation of Minerals.—In the *Australian Mining Standard* for April 5, 12, 20 and 27, 1911, Kenneth A. Mickle presents a very exhaustive statement of the conditions affecting the flotation of minerals in different solutions. Details are given of different experiments on many minerals and solutions under different conditions. It is impossible to present the details of the work, but the following conclusions will be interesting.

All minerals absorb gases.

Sulphides appear to naturally absorb CO_2.

The consequence of this is—

(1) Particles are not wholly wetted when immersed in water.

(2) Particles also tend to float when sprinkled on the surface.

(3) When immersed in water and heated, the air or gases dissolved in water will collect on the particles, and float or tend to float them to the surface.

(4) Saturated solutions of gases in water evolve gas which collects on all particles.

The Addition of Acids.—Metallic particles, such as sulphides and metals, when immersed in dilute acid solution, are not wholly wetted, but particles of rock materials become more readily wetted, and give off their adsorbed gas. When different dry minerals are sprinkled on the surface of dilute cold acid the tendency for them all is to float as in water. The minerals are more readily wetted by hot acid solutions than cold.

The Addition of Alkalis.—When different mineral particles are immersed in an alkaline solution, they tend to part with their adsorbed gases, and will not float even when heated. When sprinkled dry on the surface of the solution they remain for a few seconds only and then break through.

This is due to the property possessed by minerals of collecting gases on their surface in such quantity as to diminish the density of the product, to less than that of the liquid. By calculation from the decrease in density it is found that the amount collected on dry sulphides is sometimes half the volume necessary to float them at ordinary temperature. Given a starting point, any gas generated in or introduced into the solution will tend to collect on the surface of the particles. If acid is added, the gases will leave the gangue particles and collect on the sulphides; the latter, therefore, tend to float, and the former to remain under the surface of the liquid. When substances in the solution, such as carbonates, are attacked by acids, the gases evolved saturate the liquid, and commence to grow on the sulphides. Sulphuretted hydrogen, steam, air, and any other gases present either in the solution, introduced into it, or evolved from the minerals present, assist floatation, provided the sulphide particles have dry points on them for the gases

to become attached to. Some sulphides naturally absorb gases, and such sulphides readily float. The bulk of Broken Hill ores may be taken as examples. Other ores require preliminary treatment, so as to alter their surface to some extent. This may be done in certain cases by merely heating to 250 deg. C., or immersing in some solution which will attack the sulphides, such as nitric acid or chlorine solution. Sulphides which condense or absorb gases well will gather together in a solution as flocculent masses, while those that do not remain individually distinct, like grains of sand.

In certain cases sulphides will float in acid solution at ordinary temperature, but for the most part with dilute acid solution flotation starts at about 80 deg. C. The explanation put forward that flotation at that temperature is due to certain carbonates, such as rhodochrosite, siderite, smithsonite being attacked and giving off CO_2, does not appear to be correct, for under reduced pressure these minerals are attacked, but flotation does not take place. The presence of silicic acid and sulphur, as stated by De Bavay as necessary for a coagulum, is not necessary in many cases of successful flotation. When minerals are treated with dilute acids, and then exposed to the air, certain sulphides absorb gases on their surface. If they are alternately wetted and exposed to the air they will become entrapped by the surface film of water which they are brought in contact with. The appearance of the film is the same as that of a bubble of gas under the surface of a solution with sulphides attached. The sulphides in both cases adhere to the surface film of the bubble or free surface of the liquid, but are mainly below the surface itself. The gas obtained in all cases from experimental work and also from samples obtained from various flotation plants was CO_2, with varying quantities of nitrogen and oxygen. There is no doubt that most of this gas is generated from gangue particles in the ore, and being more soluble in the liquid than the other gases, collects readily on the surface of the sulphides.

In all experiments with sulphides carbon dioxide was obtained on exhaustion. This gas must have been present either in the form of a compound easily decomposed by heating or by reducing the pressure, or as a gas condensed on the surfaces of the sulphides, and there retained with greater force than other gases present. It was found that the carbon dioxide was the last gas to come off.

The Effect of Oils.—The effect of oils has long been known in protecting minerals from being wetted by water owing to the oiling of their surfaces, and also the affinity of oils for metals and many metallic substances. Nearly all the minerals tried could be coated with oil either in the presence or absence of water. This seems to be connected with the absorption of gases on the particles themselves, for if the particles are deprived of their gas, then as a rule they do not become coated with it. It would therefore appear that the oiling of particles is dependent on their gaseous attachments. If the particles are oiled to a minute extent, the gases previously absorbed are retained, and if a solution is heated more gas will become attached to the particles, and flotation will ensue. An oil float may be obtained without acids. Nearly all minerals will float on water after being oiled.

The flotation is apparently due partly to the lesser density of the oil and the presence of gases absorbed on the minerals or entangled in the oil. If a large quantity of viscous oil is used gases become entangled in it, and the product is oily. In course of time the particles will drop away from it, carrying down attached globules of oil. When a lesser amount of oil is used the product becomes a coherent mass like putty, showing no apparent tendency to float. On further decreasing the quantity of oil, the product becomes firmer, and breaks up into rounded pellets. With a still less quantity of oil, when shaken up with water in presence of air the product becomes flocculent, and increases in volume. This flocculation is due to the absorption of gases. With very minute quantities of oil the flocculation becomes more pronounced, and the tendency to

float becomes greater. These results may be obtained with gangue and metallic substances. Gases collect more readily on oiled than on unoiled particles.

The Effects of Acids on Oiled Particles.—The addition of acid causes a more selective action. If oiled gangue and metallic particles are immersed in water, and acid is added, then the oil will tend to leave the gangue particles and become attached to the sulphides. The action of the oil in this respect is similar to that of the gases attached to different minerals, and probably is due to the same cause.

If an ore is treated with an acid and the acid is removed, then if the particles are oiled, the oil tends to become attached to the sulphides only. This is probably due to the removal of the gas from the gangue, which prevents oiling of the same. If acid and oil are added together, the action of the oil is selective, as in the previous case. The presence of oil on their surface does not prevent the minerals being attacked by acids.

The amount of acid required for selective flotation depends on the composition of the ore, the temperature and the pressure existing at the time. Generally it may be stated that an increase of acid will give a purer sulphide product, but flotation will take place at a lower temperature, and with a lesser amount of acid, with oil and acid than with acid alone. A selective flotation can be obtained in some cases by treatment with a chlorine solution previous to the addition of oil, and without the addition of acid.

Some of the light volatile oils cause flotation in an acid solution at a lower temperature than the heavier and thicker oils. With some Broken Hill tailings a flotation takes place at ordinary temperature with sulphuric acid and a volatile oil.

Effects of Oils and Alkalis.—Flotation can be obtained in alkaline solutions when oil is used, but in every case tried the action was non-selective. On making the solution acid the action becomes selective, only the sulphides floating.

The flotation of mineral particles appears then to depend on the particles having gas attached to them becoming entangled, or caught, by the surface film of a liquid. This film may be (1) the free surface of a liquid, (2) the surface of a bubble of gas under the surface of a liquid, or (3) the surface of a bubble of gas above the surface of a liquid. Once the particles are entangled in this film they are sustained there. That a considerable weight may be supported by this surface film is shown by the experiment of floating fine-grained mineral on the surface film of water.

The Purchase of Fuel Upon a Heat Unit Basis, in Germany and Switzerland.

By John B. C. Kershaw.

Considerable attention has been given in the United States to the methods of purchasing fuel upon a heat-unit basis, and the action of the government in arranging for the application of this system of purchase to all government contracts has given a great impetus to the movement for the use of scientific methods in the purchase and control of fuel supplies.

In view of the widespread and general interest in the subject in the United States the writer has thought that the following details of the progress of a similar movement, in Germany and Switzerland, would be found interesting.

In *Germany* considerable progress has been made during the last two years. In their report for the year 1909 the committee of the Hamburg Smoke Abatement Society state that an increasing proportion of the members of the society are taking samples and having tests made of their fuel deliveries. Over 1000 fuel tests were made at the official testing laboratories of the society in Hamburg in 1909, and it is expected this total will be largely exceeded by the figures for 1910 and 1911. The chief engineer of the Hamburg Smoke Abatement Society (Herr Nies) is firmly convinced of the value of chemical and calorific tests for checking the quality of the deliveries of fuel. This

society, which had at the end of 1909 365 members and over 1300 boiler and other furnaces under its control, has arranged to undertake the regular sampling of their fuel supplies for those members who desire to be relieved of this work. This is a great step in advance, for to obtain a really fair average sample of coal or slack demands more skill and knowledge of sampling than is generally found in those who are trusted with this work, and the slow progress of the movement for more scientific control is partly due to this difficulty.

The actual testing of the fuel sample in Hamburg is undertaken by an independent fuel expert, Dr. Aufhäuser, who is head of the Thermo-chemische Prüfungs und Versuchs Austalt, or in plain English, of the Thermochemical Testing and Research Laboratory in Hamburg. The following are the details of the basis upon which the fuel is valued:

An average sample of the fuel is taken from the first or trial delivery after discharge at the consumer's works, and this is carefully tested at the official testing laboratory for moisture, ash and calorific value. The latter value is calculated on the assumption that the products of combustion of the fuel pass away in the state of vapor at 212° Fahr.—and with the aid of this result the calorific value of the combustible portion of the fuel is also calculated.

The regular samples of fuel taken to check the quality of the further deliveries are only tested for the percentage of water and ash; and from the results of these tests the calorific value is calculated by the aid of the figures obtained from the first or guarantee test. Should this calculated value deviate in either direction from the guaranteed value by more than the difference allowed by the term of the contract, an addition to, or deduction from, the price follows.

In those cases where doubt exists as to the correctness of the results, which in some cases may vary on account of a change in the calorific value of the combustible matter of the coal, the sample is submitted to all three tests, and if a different calorific value is found by the direct test, a correction is made accordingly.

It will be seen that this method of checking the fuel deliveries differs in two respects from that practiced in this country. In the first place the guaranteed calorific value of the fuel is based upon the "combustible" matter in the same, and is calculated for gasses escaping at 212° Fahr.; and in the second place, the regular samples of the fuel are only tested for moisture and ash, and the calorimeter tests are only repeated in cases of dispute.

As regards these modifications there is no doubt much to be said for the change. A fuel consists, from the practical man's point of view, of combustible and incombustible matter, the latter being made up of water and ash. If the fuel comes always from the same seam in the same pit the calorific value of the combustible portion of the coal ought to be a constant and not a variable value. The variables are the moisture and ash, and by estimating the percentage of these in each sample, and then calculating the value of the remaining combustible by aid of the value obtained in the first test, one can obtain the calorific value of the fuel actually delivered, with a minimum of time and expense.

But, this system of valuation ought to be checked, say, monthly or quarterly, by repeat tests with the calorimeter of the actual calorific value of the fuel, and the latter figure should be used until another determination is made. This will guard against errors due to the slow changes in character and quality, which undoubtedly occur in the working of some seams of coal, especially when the thickness is diminishing. It is also advisable to have some guarantee that the deliveries of fuel are from the mine and seam of coal stipulated in the contract. Mine managers often ship other coal than that bought, unless a penalty is arranged for such conduct.

As regards the assumption that the exit gases from the combustion of the fuel are always cooled to 212° Fahr, and the correction of the calorific value accordingly, this method of expressing the value is supposed to have the advantage that it

renders the calculations of the boiler efficiencies more simple, and that one can more easily judge from the calorimeter test results the actual steam-raising capacity of the fuel. As a rule, English and American calorific test results are based upon the escape of the gases at 15° C. and the value obtained in English, American and German calorific tests of fuel are, therefore, not comparable, until allowance has been made for this difference in the method of calculation. But no boiler is so efficient as a heat abstractor that the exit gases escape at a temperature of 212° Fahr. The German test results, therefore, still require correcting for the difference between 212° Fahr. and 560° Fahr., or 600° Fahr., which is generally the temperature of the escaping gases, and the simplifying of the calculations required when estimating boiler efficiencies is thus more apparent than real.

Below are a group of test results calculated on this basis obtained at the official fuel testing laboratory of the Hamburg Smoke Abatement Society and taken from the last annual report of the society:

| | | Percentage of | | Calorific values in centigrade units. | |
	Moisture.	Ash.	Combustible.	Raw coal.	Combustible.
Durham coal.........	1.6	2.2	96.2	7951	8273
	4.6	15.8	79.6	6339	7999
	2.6	6.9	90.5	7350	8139
Northumberland coal.	1.4	13.4	85.2	6907	8115
	5.4	8.7	85.9	6840	8001
	12.1	5.5	82.4	6413	7871
Yorkshire coal.......	3.5	7.3	89.2	7145	8034
	6.4	4.8	88.8	6987	7912
	3.7	3.3	93.0	7526	8116

The above calorific tests require multiplying by the factor 1.8 to obtain the corresponding values in British thermal units, and it will be seen that there is a margin of 402 Centigrade units (723 British thermal units) between the highest and lowest results when based on pure coal or combustible. This difference proves that even after the two variables, moisture and ash, have been eliminated from the coal, considerable variation exists in the calorific value of the coal mined in different collieries and seams in the North of England and that the Hamburg system of valuation, if adopted, must be used with discretion and common sense.

As regards the progress of scientific methods of controlling fuel supplies in *Switzerland* the state authorities in that country established in 1907 at a cost of $10,000 a well-equipped fuel testing laboratory at Zurich, and have since 1908 based all contracts for the supply of fuel to the State railways upon calorific tests.

The principle employed here is to fix an average heat value for the fuel being delivered (in this case German and French briquetted fuel) and to deduct from or add to the price a proportionate sum for every 50 heat units (90 British thermal units) by which the calorific value as delivered differs from this standard. An illustrated description of the State fuel testing laboratory at Zurich appeared in the issue of the *Engineer* for June 12, 1908, and it was there stated that over 3000 samples of fuel had been tested in the first year of its existence.

Most of these represented German fuels, for Switzerland has to import all her coal, at an annual cost of $15,000,000. The remaining samples were of Belgian, French and English fuels.

The director of the laboratory at Zurich, Professor Coustam, has stated that this extensive examination of imported fuels in Switzerland, both for the State and private consumers, has tended to enlighten the buyers as to the qualities and economic value of the fuels from the various countries and collieries and has also contributed toward the classification of fuels according to their heating power or value.

In a recent letter to the author of the present article Professor Coustam confirms this view, and asserts that fuel testing in Switzerland is centralized at the State Fuel Testing Laboratory where practically all fuel for railway and gas-making purposes is tested regularly—i. e., a sample being taken from every twentieth car-load of coal as it arrives. In 1909 4000 samples and in 1910 4400 samples were tested calorimetrically, besides about 1000 samples of gas-coal tested for moisture, ash, volatile matter and sulphur.

All parties concerned (producers *and* consumers) appear to be satisfied with the methods of fuel testing, the former agreeing that they are thereby protected from unjustified complaints. The results of the tests are compiled monthly, and are communicated to the sellers and buyers, the monthly mean serving as basis for payment.

Liverpool, England.

Tests for Fibrous Insulating Materials.

Next to mechanical strength, the most important factor in determining the value of different types of electrical generating machinery is insulation. The size and life of dynamos and motors depends not so much on the actual current-carrying capacity of their conductors as on commutation, and the resistance of the insulating materials employed to the disintegrating action of heat, in the first place, and humidity in the second.

A dry heat need obviously only be limited in intensity by the point of combustibility of the substances employed, and it has naturally always been the aim of electrical manufacturers to employ substances as incombustible as possible. A test for inflammability under dry heat is easily contrived, and has not therefore been considered so important as one under the more onerous conditions of heat and moisture simultaneously applied.

Tests under these circumstances and the data gathered form the subject of a paper presented to the British Institution of Civil Engineers by Mr. W. Pollard Digby. This paper more particularly alludes to tests on the three well-known substances, "Press-spahn," "Leatheroid" and "Empire" cloth.

The test employed was to expose strips of the substance to be investigated, of known area and weight, to the action of a vapor bath having a temperature of 167° F. (75° C.) for periods aggregating 1 hour and 40 minutes, the weight and insulation resistance per square inch being taken every 20 minutes. The results of the test go to show how widely thin "Empire" cloth (between 4 and 5 mils thick) varies in its properties. Increases in thickness for this substance were obtained from 1.7 to 11.1 per cent., and increases in weight of from 5.9 to 7.8 per cent., the insulation resistance at the end of the test varying from 10.1 to 11.2 megohms. Thicker material (from 6 to 6.9 mils) increases in weight of from 2.6 to 6.6 per cent., were noted, and the insulation resistances on three samples were as high as 5,000 megohms per square inch. These tests showed the value of using a single thickness of, say, 12 or 14 mil material, rather than an equivalent thickness of two or more sheets of thinner material.

Press-spahn or fuller-board, unvarnished, gave, on the whole, poor results. The more satisfactory samples varied from 1.2 to 40 megohms, and other results were as low as .0001 of a megohm. The thickness seemed to make little difference. The conclusion come to was that "Press-spahn" was unsuited for any but small low voltage machines.

Leatheroid gave more satisfactory results, the poorer registering a resistance after test of 10.8, 7.7 and 4.9 megohms per square inch, while six out of nine samples gave 40 megohms per square inch, with increase in thickness not exceeding 3.4 per cent. The thicker samples (over 10 mils), however, were not so uniformly good, only three out of twenty having insulation resistances of more than 1,000 megohms per square inch. As a potential specification for this substance, the author permits the use of leatheroid, which, after the vapor bath test at 167° F. for 100 minutes, has an average insulation-resistance of over 20 megohms per square inch and an increase in weight and thickness not exceeding 5 per cent. This substance compares very favorably with press-spahn, as, for any given

increase in weight and thickness the moisture absorbed lowers the insulation resistance of leatheroid to a much less degree than that of press-spahn.

The function of these materials (press-spahn and leatheroid) is generally described as mechanical, i.e., they are used to protect the actual insulating sheaths of molded mica or woven material from abrasion in the slots. At the same time it is obviously advantageous that the most durable and non-hygroscopic material should be used.

Data on dielectric strength were only available on one specimen of leatheroid, which was tested by a manufacturing firm. The volts per mil thickness were here found to increase up to 15 mils in undried, and 21 mils in dried leatheroid, after which they apparently fell off. The highest recorded was 475 volts (on 8 mil material), and the lowest 229 volts on 21 mil.

As a result of thirty-two tests it was found that "leatheroid" might contain up to 5.5 per cent., and "Empire" cloth up to 2.96 per cent. of moisture and volatile matter, the figures for vulcanized fibre ranging up to 2 per cent.

The paper concludes with a suggestion that a formula expressing the minimum permissible resistance for motors and generators after heat-run should read: $R = Kv \div s$, where $v =$ voltage of machine, $s =$ estimated net iron-area of one slot, multiplied by the number of slots, and $K =$ a constant (not less than 15). This formula would differentiate between high-speed machines with small surface and low-speed generators of large diameter.

NOTES.

The Linde Air Products Company, of Buffalo, N. Y., have just shipped a complete oxy-acetylene cutting plant arranged for six operators to the War Department for use in conjunction with the salvage of the U. S. S. Maine in Havana harbor. Much of the work involved is the cutting of 9-in. armor plate in addition to the usual structural shapes. It is interesting to note that similar apparatus to this was used with signal success in connection with the removal of the debris of the Quebec bridge.

Dings electromagnetic separators are built in low, medium and high intensity types designed for different kinds of work. The low intensity type is useful for separating metallic iron from non-magnetic material. Those of medium intensity are used for the separation of roasted sulphide ores, magnetic iron ores, etc., while the high intensity machines are applied mainly to the separation of wolframite, monazite, rutile and minerals with very feebly magnetic constituents.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in March, 1911, was 68. The total tonnage milled during that period was 2,005,526 tons of ore; 9911 stamps were in operation with an average duty of 7.62 tons per 24 hours. Tube mills in commission numbered 209. The yield for the month was 676,065 fine ounces gold. This report shows an increase of about 200,000 tons of ore milled and an addition of 47 tube mills in commission as compared with one year ago.

In the Annual Report of Phelps, Dodge & Co. appears a brief statement of the experiments on dust settling at the Copper Queen Smelter. During 1910 these experiments were carried to conclusion, and it was shown beyond question that it is possible to settle a large part of the finest dust in a comparatively short but wide chamber, in which the velocity of gases is reduced to between two and two and a half feet per second. The dust chambers are, therefore, being reconstructed by sections. The first section will handle the smoke from two blast furnaces. It is 1300 square feet in area, and it is thought this will be sufficient to settle most of the fine dust. In order to reduce the amount of gases passing through the main chamber a separate flue is being built for the converters. A calcining and reverberatory smelting plant is being built to treat fine ores and flue dust. This will be completed by the end of this year. It will consist of two reverberatories with boilers built in the furnace flues to utilize waste heat; six McDougall roasting furnaces; dust chambers, flues and stack.

Filter leaves or clarifying leaves may be bound with iron wire covered with P. & B. paint instead of the usual method of sewing them together at intervals. The wire is bound around the frame outside of the cloth and then fastened with staples passing over the wire on one side of the leaf and clinching over a wire on the opposite side. The scheme may have some advantage in cases where the threads used in sewing tend to break when reverse pressure is applied to discharge the cake.

Ferric iron may be precipitated quantitatively by a 6 per cent solution of nitrosophenylhydroxylamine, according to Biltz and Hodtke in *Zeit. Anorg. Chem.*, 1910, 66, 426-431. An excess of acid does not interfere with the precipitation, and iron may be separated from nickel, chromium and aluminium in this way.

A new classifier designed by G. A. Overstrom makes use of a rising current of water in successive sorting columns for the purpose of separating the specifically heavier particles into certain grades. The slime is removed first, and in each succeeding sorting column the rising current of water is so adjusted that successively finer grades will be raised out of the pulp, while the heavier settle to the next sorting column. The classifiers may be arranged in succession to receive and treat the specifically heaviest grade from the next preceding classifier. The system requires little clear water, as the overflow from each classifier is practically clear and is used in the succeeding machine.

The cost of silver production at Cobalt, Ontario, Canada, is indicated in the fourth annual report of the La Rose Consolidated Mines Company. During the seven months covered by the report a total of 2,569,905 oz. silver were produced at a cost of 19.11 cents per ounce. Of this quantity 401,221 oz. were produced by concentration, representing a recovery of 74.61 per cent. As a result of the large quantity of cobalt offered from this district the market is overstocked, and all payments for this metal have been discontinued.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

405,824, June 25, 1889, Turner D. Bottome, of Hoosick, New York, assignor to John B. Tibbits, of same place.

Gold and silver. The ore is reduced to powder and blown by steam into an atmosphere of chlorin and atomized water, the metal being thereby converted into chlorid. The chlorinated ore falls through sprays of sodium thiosulfate, to dissolve out the chlorids. The undissolved residue is filtered out and the solution is electrolyzed between electrodes of carbon and copper, the gold and silver being deposited on the cathodes.

407,386, July 23, 1889, and 410,228, Sept. 3, 1889, Jacob C. Wiswell, of Medford, assignor by Direct and Mesne Assignments, to the Electric Gold and Silver Chlorination Company, of Boston, Massachusetts.

Produces mercuric chlorid solutions for separating gold and silver from ores. A sodium chlorid solution, to which is added ammonium chlorid or hydrochloric or sulphuric acid, is electrolyzed in a tank of wood between anodes of carbon and cathodes of zinc. The resulting solution is delivered into a second electrolytic cell containing an anode of mercury and a cathode of carbon. The mercuric solution produced in this cell is employed to free the precious metals from the gangue.

415,576, Nov. 19, 1889, Werner Siemens, of Berlin, Germany, assignor to Siemens & Halske, of same place.

Copper, zinc, etc. Lixiviates the ore with a solution of ferric

sulfate, chlorid or cyanid, which dissolves the metal and is reduced to the ferrous form. Electrolyzes the resulting solution at the cathode, depositing the non-ferrous metal; then reoxidizes the residual solution at the anode to the ferric form. The operation may be carried out in one cell or in a number of cells arranged in cascade. The single cell is divided into two compartments by a cup-shaped diaphragm. The cup contains an anode of carbon, platinum or lead, and receives one leg of a syphon, which extends nearly to its bottom. The outer compartment contains a cathode-plate and a liquid supply-tube. The cup communicates with the outer compartment through a hole near its top. In the modification employing a series of cells, the anode and cathode compartments of adjacent cells are connected by syphons. To extract copper, the ore, for example, pulverized pyrites, is roasted until the iron therein is entirely oxidized and is lixiviated with ferric sulfate or chlorid, the resulting solution containing cupric and ferrous salts. Some free acid is added to increase the conductivity and the liquid is electrolyzed at the cathode, the copper being deposited. The solution, becoming lighter, rises in the cathode compartment and, when but one cell is used, flows through the opening at the top of the cathode-cup into the anode compartment. Here the ferrous salt in solution is reoxidized into the ferric form. In the series-cell construction, the two solutions pass successively through the cathode and anode compartments of all the cells, to secure complete deposition of the copper and oxidation of the iron. The depleted liquid from the last cathode compartment is returned to the first anode compartment. The potential difference used is 0.7 volts, and the cathode current-density 25 amperes per square meter. Zinc is similarly extracted from slightly roasted blende, the necessary potential-difference being nearly double that for copper.

415,738, November 26, 1889, Hermann A. Seegall, of Berlin, Germany.

Copper, zinc, tin. Lixiviates the ores first with ferric chlorid and then with a solution capable of dissolving the residual cuprous chlorid in the ore, for example a halogen salt or acid, or sodium hyposulfite. The second solution, plus ferrous chlorid, may be electrolyzed in an open cell, or it may be electrolyzed at the cathode in a diaphragm cell with ferrous chlorid in the anode compartment. Examples: 1. Lixiviates copper and zinc sulfid, or an alloy of the two, with ferric chlorid slightly acidulated with hydrochloric acid, until the iron is reduced to the ferrous form, the zinc dissolved as chlorid and the copper in the residue converted into cuprous chlorid and a little spongy copper. The copper is filtered and electrolyzed to recover the zinc. Copper cuttings or filings are added to the residue and it is lixiviated with ferric chlorid, slightly acidulated with hydrochloric acid and containing calcium or soldium chlorid to dissolve the cuprous chlorid. The solution is filtered to remove the freed sulfur and is electrolyzed to recover the copper. 2. Copper tin scrap is lixiviated with ferric chlorid slightly acidulated with hydrochloric acid. The tin dissolves and the solution is filtered off and electrolyzed to recover it. The residual cuprous chlorid is recovered as before. A series of electrolytic cells may be employed, the solutions passing successively through the several cathode and anode compartments, that of the last cathode compartment being returned to the first anode compartment.

418,134, Dec. 24, 1889, Henry Forbes Julian, of Johannesburgh, Transvaal, South Africa, assignor to Flavien Ernest Lezard and James Joseph Coghlan, Trustees.

Gold and silver. The ore is pulverized and agitated in a closed vessel with a solution containing bromin, chlorin or iodin or their salts. A pressure of from 60 to 80 pounds per square inch is maintained in the vessel by air or steam. When the gold has been dissolved as a haloid and the silver converted into an insoluble haloid, mercury or sodium amalgam is added to the semi-liquid mass, which is again agitated, amalgamating the gold. Or the gold in solution may be precipitated. The

tailings are then run through a series of electrolytic cells, in cascade, having inclined cathode bottom-plates of amalgamated copper covered with mercury and depending anodes of carbon, platinum or lead peroxid. The remaining gold and silver are thereby dissolved and redeposited in the mercury.

452,125, May 12, 1891, Werner von Siemens, of Berlin, Germany, assignor to Siemens & Halske, of same place.

An apparatus for effecting the step of initial lixiviation in the process of his patent 415,576 (supra). The vessel is a long trough, or a series of connected troughs arranged zigzag, containing parallel shafts carrying paddles and a heating pipe.

PERSONAL

Mr. H. Foster Bain has returned to San Francisco from a trip to Colorado and will soon go to Alaska.

Mr. Thomas A. Dickson, formerly with the Colorado Iron Works Company, Denver, has gone to Goldfield, Nev., to take charge of the refinery of the Goldfield Consolidated Mines Company.

Mr. C. A. Filteau has returned to Denver from San Jose de Gracia, Sinaloa, Mexico, where he has been in charge of cyanide operations of Cia. Minera Jesus Maria y Anexas.

Mr. Franklin Guiterman, general manager of the American Smelting & Refining Company in Colorado, has left for several months' stay in Europe. He will study the progress in electric smelting with particular reference to zinc reduction.

Mr. Charles Janin, of San Francisco, was in Denver recently on a professional trip which took him through Arizona and Colorado.

Mr. William F. Kett has been appointed manager of the Mountain Copper Company, California, in place of Lewis T. Wright, who has resigned.

Mr. L. H. Norton, of Silverton, Colo., was in Denver recently on professional business.

Mr. Cooper Shapley has been appointed superintendent of the Oro Grande Company at La Luz, Guanajuato, Mexico.

Mr. George M. Taylor has been appointed general manager of the milling department of the Portland Gold Mining Company, Colorado, vice J. B. Daniels, deceased.

Mr. J. C. Welch has resigned his position as smelter superintendent of the East Butte Copper Mining Company, Butte, Mont.

Mr. Lewis T. Wright has retired from the management of the Mountain Copper Company, California, and will take up his residence in London, at least for a time.

BOOK REVIEWS.

Die Metallurgie des Zinns.—Mit spezielle Berücksichtigung der Electrometallurgie. (Band XXIX. Monographien über angewandte Electrochemie.) By Dr. **Hans Mennicke,** chemical engineer. 8vo., 196 pages, 40 illustrations, price 10 marks, bound 11.50 marks (retail price in New York $3.75). Halle a. S.: Wilhelm Knapp.

The ordinary processes of extraction of tin from ore are but briefly treated; Schnabel's "Handbook of Metallurgy" is more satisfactory on this part of the subject. The chapter on obtaining tin from 'tin skimmings, dross, and other chemical wastes is interesting and contains much new information. A 43-page chapter on the detinning of tin-scrap is probably the first publication of a description of some commercial plants. A 25-page chapter on the refining of impure tin, chemically and electrolytically, is most acceptable. Several short chapters follow on electroplating tin, production and consumption of tin; finally, there are 40 pages containing the full specifications and claims of important patents concerned with the metallurgy of tin. The whole is a useful and valuable contribution to the scanty metallurgical literature of an important and interesting metal.